PB-216 249

FEASIBILITY STUDY ON THE APPLICATION OF VARIOUS GROUTING AGENTS, TECHNIQUES AND METHODS TO THE ABATEMENT OF MINE DRAINAGE POLLUTION. PART IV. ADDITIONAL LABORATORY AND FIELD TESTS FOR EVALUATING AND IMPROVING METHODS FOR ABATING MINE DRAINAGE POLLUTION

Halliburton Company
Duncan, Oklahoma

15 April 1968

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Monongahela River Mine Drainage Remedial Project

Upper West Fork River Sub-basin

West Virginia

FEASIBILITY STUDY ON THE APPLICATION OF VARIOUS GROUTING AGENTS, TECHNIQUES AND METHODS TO THE ABATEMENT OF MINE DRAINAGE POLLUTION

DRAFT
FINAL REPORT
PART IV

ADDITIONAL LABORATORY AND FIELD TESTS FOR EVALUATING AND IMPROVING METHODS FOR ABATING MINE DRAINAGE POLLUTION

Prepared for the

U.S. DEPARTMENT OF THE INTERIOR

FEDERAL WATER POLLUTION CONTROL ADMINISTRATION

Details of illustrations in
this document may be better
studied on microfiche.

By

HALLIBURTON COMPANY
DUNCAN, OKLAHOMA

April 15, 1968

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The research upon which this publication is based was performed pursuant to Contract No. 14-12-31, with the Federal Water Pollution Control Administration, Department of the Interior.

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Halliburton COMPANY
DUNCAN, OKLAHOMA 73533

April 15, 1968

Mr. John R. Hyland, Project Officer
Monongahela River Mine Drainage Remedial Project
401 Methodist Building
11th and Chapline Streets
Wheeling, West Virginia 26003

Re: Feasibility Study on the Application of Various Grouting Techniques and Methods to the Abatement of Mine Drainage Pollution

Dear Sir:

In accordance with the terms of the Department of the Interior, Federal Water Pollution Control Administration Contract No. 14-12-31, we submit herewith the final draft of the Part IV Report for the subject feasibility study.

We appreciate the cooperation that has been accorded us, both by your staff and by others of the Federal Water Pollution Control Administration and the U.S. Department of the Interior.

Sincerely yours,

Ray W. Wenzel

Ray W. Wenzel
Project Manager

UNITED STATES DEPARTMENT OF THE INTERIOR
FEDERAL WATER POLLUTION CONTROL ADMINISTRATION
633 Indiana Avenue, N.W.
Washington, D.C. 20242

ADDITIONAL LABORATORY AND FIELD TESTS FOR EVALUATING AND IMPROVING METHODS FOR ABATING MINE DRAINAGE POLLUTION

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Mr. Paul W. Hornor, Consulting Mining Engineer, Hornor Brothers Engineers of Clarksburg, West Virginia was utilized for survey of mine properties, site mapping and general consultation.

I. INTRODUCTION

The initial phase of this project under FWPCA Contract 14-12-31 involved the studies and investigations performed in the exploration of drift mines in the Upper West Fork River Sub-basin, West Virginia. Included as a part of the study was the application of various grouting agents and techniques to the abatement of mine drainage pollution.

Inventories made by the Monongahela Enforcement Project in 1965 provided the basis for selection of drift mines. A total of 228 drift mines or openings were found during the survey.

The Part I report discusses the geology and hydrology of the project area, suggested materials and techniques to be used in remedial action, as well as findings of the survey.

A later report for Part II discussed the designation and further investigation of 20 selected mine sites from the initial study which appeared to be mines in which the abatement techniques could be utilized. Also included in that report is a discussion of a remedial application plugging one drift opening utilizing techniques and materials set forth in the Part I report.

From the twenty mines listed in the Part II report, three mines were selected and specific plans and specifications were written covering remedial construction work to be accomplished at each of the sites. These are included in the report for Part III of the project.

A separate report was furnished to the Project Officer on an investigation of the effect of various chemical additives on the oxidation of pyrite.

Part IV of the project is a modification of the original contract for the purpose of conducting additional field tests on methods and materials not previously evaluated in this project. The Part IV program includes an experimental test program, a seismic survey and study, and field construction tests and evaluation.

The experimental test program was conducted for the purpose of determining improvements in techniques and materials previously used for remedial applications as well as consideration of different techniques which might be evaluated in field tests. This specifically involved tests to select an economical grout slurry with a fast set time, evaluation and improvement of Expendable Grout Retainers, tests of small retainers and subsequent tests of full scale retainers in a simulated mine opening, and the design and testing of plugs for sealing auger holes. Also included is an evaluation of grouting preplaced aggregate plugs and remote bulkhead building using a cementitious material.

The seismic survey and study included in this part of the project is for the purpose of locating hidden mine openings or thin high wall sections by seismic methods.

The field construction studies were conducted for the purpose of testing and evaluating methods and materials for remedial construction which had not previously been applied during this project. Included in the field operations were the following tests:

1. Gob disposal on Mine No. 62-009 by conveying in a water mixture.
2. Remote wall preparation prior to plugging of openings on Mine No. 40-016.
3. Pneumatically placing aggregate plugs in mine drifts in Mine No. 40-016.
4. Grouting of preplaced aggregate to form plug in drift openings of Mine No. 40-016.
5. Placement of Horizontal Grout Curtain to prevent or minimize the percolation of water into area of auger holes on Mine No. 62-009.
6. Plugging of auger holes utilizing wooden plugs and cloth retainers on Mine No. 62-009.
7. Installation and monitoring of water-level recorder on Mine No. 14-042A.

II. EXPERIMENTAL TEST PROGRAM

Summary

The objectives of the test work conducted in the Duncan, Oklahoma Research facilities were:

(1) To evaluate present remedial methods for the purpose of improving the design and gaining economical advantages thereof.

(2) To test and evaluate various other methods and materials indicated as feasible for plugging drift openings and auger holes to extend and improve these methods.

In order to accomplish this, the work was conducted in a number of phases. A study was conducted on various grout slurries to find a fast setting, economical slurry which would enable the placement of the plug in a mine opening in one day's operation. The slurry also had to meet a certain strength requirement in order to accomplish the operation as desired.

Another phase of the test work involved the improvement of the Expendable Grout Retainer. In this investigation, various fabrics were considered and tested. The selected fabric was made into small-scale retainers and further tests were made. A full-scale test of the retainer type plug was made in a simulated mine opening as a test of the improved retainer and the sealing slurry. Incorporated in the full-scale test was complete instrumentation to provide information on the characteristics of the retainer type plug when used in a drift opening.

As part of this phase, tests were conducted on the bond strength of the plastic pipe to be used, as well as bond strength of the retainer to the surrounding wall and to other retainers.

A study was made to develop means of plugging auger holes. Two different types of plugs were developed and tested, either of which appeared to be suitable for field application and evaluation.

A series of tests was made both on small-scale and full-scale to provide confirmation of work planned to be accomplished in the field construction. The tests consisted of placing aggregate pneumatically for building a bulkhead and consolidation of aggregate by grouting methods. Full-scale tests were also conducted on pneumatically placing aggregate in an Expendable Grout Retainer and subsequently consolidating with cement grout.

Experimental work was conducted on both small and full-scale basis in building a bulkhead remotely by spraying a cementitious material.

The test program was conducted by personnel from the Chemical Research and Development Department, Mechanical Research and Development Department and Bulk Materials Department.

A. Grout Slurry Evaluation

A grout slurry may be used in several different applications. One use is in filling an Expendable Grout Retainer or Auger Hole Plug. Another application could be in filling an area between bulkheads to plug a tunnel or drift opening. Still another use would be in remotely building a bulkhead in a mine drift.

In order to establish criteria for the slurry in each application, specifications were written.

The first application considered was that of filling Expendable Grout Retainers or Auger Hole Plugs. Expendable Grout Retainers are cloth containers which, when filled with slurry, conform to the shape of the opening in which they are placed.

Slurry used in filling the Expendable Grout Retainers or Auger Hole Plugs would be selected in accordance with the following specifications:

(1) Slurry should be economical.

(2) Slurry should be fast setting.

(3) Slurry should obtain a minimum compressive strength of 10 psi at 70° F. within one hour set time after placement, assuming one hour would be used for mixing and placing.

(4) Slurry should have a minimum compressive strength of 300 psi at 70° F. after 48 hours.

(5) Slurry should be able to bond to coal.

(6) Slurry should remain in a pumpable condition for one hour after mixing. Viscosity should not exceed 35 poise.

Specification number (3) was determined by the load necessary for a retainer to support when successive retainers are placed on top, allowing a safety factor of 3. This would also allow the placement of sufficient retainers in one day to completely plug a drift opening. Specification number (4) is based on the assumed overburden expected on the average mine site.

The slurries investigated for use in Expendable Grout Retainers or Auger Hole Plugs were composed of the following materials:

(1) Cement: Portland type, conforming to ASTM Type 1 specifications, ASTM C150-66.

(2) Fly Ash: Fly Ash is a product of combustion of pulverised coal, consisting chiefly of alumina, silica, iron oxide, lime and magnesia, and shall conform to ASTM C350. Source - Monongahela Power Co., Rivesville, West Virginia.

(3) Calcium Chloride: Conforming to specifications as set forth in ASTM D98-59, Type 1 or 2.

(4) Gypsum Cement: Hemihydrate form conforming to specifications as set forth in ASTM C59-50, Gypsum Molding Plaster. Materials used in tests were Bestwall, manufactured by Georgia Pacific Corp., Bestwall Gypsum Division; and Bondcrete, manufactured by U.S. Gypsum.

(5) HR-7: A dispersant type lignin retarder available from Halliburton Services, Weston West Virginia.

Tests were conducted to determine the desired amounts of each ingredient to meet the established specifications for setting time and compressive strength. The amount of each ingredient was determined for the quantity required to produce a cubic yard of slurry. Costs were based on purchase prices at Clarksburg, West Virginia, but did not include any charges for blending, mixing, pumping or transportation.

Table I gives information on six slurries tested. Slurry No. 5 did not meet the compressive strength specification established, but it was tentatively selected for use in the full-scale tests because of its low cost and other desirable qualities. It was later modified by decreasing the amount of water to obtain Slurry No. 6. However, although other tests were made using Slurry No. 5, results would be applicable to Slurry No. 6 since it is the same slurry with less water.

TABLE I

GROUT SLURRIES FOR EXPENDABLE GROUT RETAINERS

Slurry No.	Materials per Cubic Yard of Slurry						Slurry Weight Lbs./Cu. Yd.	Cost/Cu.Yd.
	Pounds					Gallons		
	Cement	Fly Ash	$CaCl_2$	Gypsum	HR-7*	Water		
1	2068	---	83	---	---	124	3078	$30.82
2	1081	851	80	---	---	120	2916	19.66
3	712	---	--	1068	10.7	128	2835	26.53
4	522	522	--	696	10.5	125	2781	20.66
5	345	692	--	692	10.4	125	2754	18.65
6	430	860	--	860	18.2	105	3025	24.50

Slurry No.	Slurry Description***
1	100 parts Cement + 4 parts Calcium Chloride**; 50 parts water
2	56 parts Cement, 44 parts Fly Ash, 4 parts Calcium Chloride; 50 parts water
3	40 parts Cement, 60 parts Gypsum Cement + 0.6% HR-7; 60 parts water
4	30 parts Cement, 30 parts Fly Ash, 40 parts Gypsum Cement + 0.6% HR-7; 60 parts water
5	20 parts Cement, 40 parts Fly Ash, 40 parts Gypsum Cement + 0.6% HR-7; 60 parts water
6	20 parts Cement, 40 parts Fly Ash, 40 parts Gypsum Cement + 0.6% HR-7; 41 parts water

* HR-7 is a trade name for a dispersant type lignin retarder marketed by Halliburton

** 4 parts of Type 1 or 3.2 parts of Type 2

*** Parts given are parts by weight.

TABLE I (Continued)

GROUT SLURRIES FOR EXPENDABLE GROUT RETAINERS

Slurry No.	Thickening Time Hrs.	Thickening Viscosity Poises	Compressive Strength - Psi 2 Hrs.	2.5 Hrs.	3 Hrs.	18 Hrs.	24 Hrs.	48 Hrs.	72 Hrs.
1	1	(29)	14	---	102	---	2365	---	---
2	1	(19)	Firm	12.5	---	495	---	---	---
3	1	(17)	285	---	---	---	475	---	---
4	1	(19)	32	---	---	---	200	363	400
5	1	(8)	55	---	---	---	147	233	250
6	1	(25)	215	---	---	---	325	---	---

Slurry No.	Slurry Description***
1	100 parts Cement + 4 parts Calcium Chloride**; 50 parts water
2	56 parts Cement, 44 parts Fly Ash, 4 parts Calcium Chloride; 50 parts water
3	40 parts Cement, 60 parts Gypsum Cement + 0.6% HR-7; 60 parts water
4	30 parts Cement, 30 parts Fly Ash, 40 parts Gypsum Cement + 0.6% HR-7; 60 parts water
5	20 parts Cement, 40 parts Fly Ash, 40 parts Gypsum Cement + 0.6% HR-7; 60 parts water
6	20 parts Cement, 40 parts Fly Ash, 40 parts Gypsum Cement + 0.6% HR-7; 41 parts water

* HR-7 is a trade name for a dispersant type lignin retarder marketed by Halliburton

** 4 parts of Type 1 or 3.2 parts of Type 2

*** Parts given are parts by weight.

The Heat of Hydration of each slurry and the Heat Capacity of a set specimen are shown in Table II for four selected slurries. This test was intended to show the temperature rise that could be expected while plugging an opening using each slurry. It can be seen that Slurry No. 5 exhibits less temperature build-up and would therefore be less subject to cracking due to heat hydration.

Expansion bars were made on several selected slurries to insure a slurry which would have a good bond with the four sides of an opening. These results are shown in Table III. It can be noted that Slurries No. 4 and 5 exhibited the most expansion in these tests and would therefore provide a tighter bond to surrounding formations.

Slurry No. 5 was modified to Slurry No. 6 by reducing the water content from 60 to 41 parts water. This produced a denser slurry which met the specifications. Tests were then conducted on Grout Slurries 5 and 6 to determine the effect of mixing water temperature and amount of retarder added. These test results are shown in Table IV. It can be noted that the compressive strength is approximately the same with temperature of mixing water at 45° F. and 70° F. It was also determined that a minimum of 0.4% retarder is necessary to allow sufficient pumping time for the slurry.

When Slurry No. 5 was modified to Slurry No. 6 as a result of information gained from the full-scale tests, the cost increased from $18.65 to $24.50 per cubic yard because of the extra solid materials required. This occasioned a review of other slurry test results to find a less expensive selection. Slurry No. 4 met the required specifications and had an expansion second only to Slurry No. 5. The cost is $20.66 per cubic yard. It is therefore recommended that Slurry No. 4, composed of 30 parts Cement, 30 parts Fly Ash, 40 parts Gypsum Cement plus 0.6% HR-7 retarder and 60 parts water, be selected as the slurry for use in filling grout retainers or auger hole plugs.

It is also recommended that Slurry No. 5, composed of 20 parts Cement, 40 parts Fly Ash, 40 parts Gypsum Cement, 0.6% HR-7 Retarder and 60 parts water by weight, be accepted as an alternate selection with the understanding that its compressive strength of 233 psi in 48 hours does not meet the specifications. This slurry would require longer to attain 300 psi compressive strength as required, but is acceptable from other standpoints and would provide a cost savings of $2.01 per cubic yard over Slurry No. 4.

TABLE II

HEAT CAPACITY AND HEAT OF HYDRATION

Slurry No.	Heat Capacity Calories/gm/°C.	Maximum Temp. Reached	Time Hrs.	Heat of Hydration BTU/Lb. of Slurry
1	0.414	238° F.	5.0	65.29
2	0.441	202° F.	11.9	63.26
5	0.488	131° F.	16.7	34.16
11	0.436	186° F.	10.0	64.52

Slurry No.	Description of Slurry*
1	100 parts Cement, 4 parts Calcium Chloride; 50 parts water
2	56 parts Cement, 44 parts Fly Ash, 4 parts Calcium Chloride; 50 parts water
5	20 parts Cement, 40 parts Fly Ash, 40 parts Gypsum Cement, 0.6% HR-7; 60 parts water
11	56 parts Cement, 44 parts Fly Ash, 2 parts Bentonite; 63 parts water

* Parts given are parts by weight.

TABLE III

EXPANSION OF VARIOUS SET SLURRIES

Slurry No.	Percent Expansion				
	15 Days	30 Days	45 Days	60 Days	75 Days
4	.173	.479	.578	.831	.982
5	.182	.740	.916	1.112	1.112
8	.078	.105	.121	.137	.148
9	.088	.113	.128	.143	.155
11	.055	.075	.09	.103	.115

Slurry No.	Description of Slurry**
4 *	30 parts Cement, 30 parts Fly Ash, 40 parts Gypsum Cement; 60 parts water
5 *	20 parts Cement, 40 parts Fly Ash, 40 parts Gypsum Cement; 60 parts water
8	56 parts Cement, 44 parts Fly Ash, 2 parts Bentonite, 11.5 parts Salt, 63 parts water
9	56 parts Cement, 44 parts Fly Ash, 2 parts Bentonite, 23 parts Salt, 63 parts water
11	56 parts Cement, 44 parts Fly Ash, 2 parts Bentonite, 63 parts water

* Contained 0.6% HR-7 Lignin Retarder

** Parts given are parts by weight

TABLE IV

EFFECTS OF TEMPERATURE, MIXING WATER AND RETARDER

Slurry*	Per Cent HR-7 Retarder	Set Time-Min. 45° F.	Thickening Time-Min. 45° F.	Compressive Strengths - Psi		
				1 Hour**		24 Hours**
				45° F.	70° F.***	45° F.
5	0.0	5				
6	0.0	2				
5	0.2	20	--	175	---	163
6	0.2	17	--	620	---	615
5	0.3	35	--	165	145	125
6	0.3	20	--	350	290	405
5	0.4	--	39	140	83	---
6	0.5	--	46	435	475	420

* Slurry No. 5:

20 parts Cement, 40 parts Fly Ash, 40 parts Gypsum Cement, 60 parts water. Slurry Weight: 13.6 lbs./gal.

Slurry No. 6:

20 parts Cement, 40 parts Fly Ash, 40 parts Gypsum Cement, 41 parts water. Slurry Weight: 15.0 lbs./gal.

Note: Water Temperature: 45° F.
Curing Temperature: 45° F.
Parts given are parts by weight.

** Time from start of mixing.

*** 70° F. data from previous tests is shown for comparison.

Another application for grout slurry would be in filling the space between bulkheads placed in a mine drift.

Slurry used in areas between bulkheads should have the characteristic of expanding during the setting period to assure a good bond around the plug. It should also have a minimum compressive strength of 300 psi at 70° F. after 48 hours, and be economical to use.

The slurries investigated for use in mine drifts between bulkheads included most of the same materials as used in tests on grout slurries for Expendable Grout Retainers. In addition, the following three materials were used:

(1) Salt (Sodium Chloride): An evaporative type crystaline compound of high purity available in granulated form, conforming to specifications for Type II as set forth in ASTM D632-66.

(2) Bentonite: Wyoming type (Sodium Montmorillonite) conforming to specifications as set forth in API STD 10A, API "Specifications for Oil Well Cements and Cement Additives", Section 3.

(3) Gob: That part of the mined material, either coal or other materials, that is not marketable and is therefore wasted. Material must be screened and graded prior to use as a slurry component.

Tests were conducted to determine the desired amounts of each ingredient to produce the desired results. The amounts required to make a cubic yard of slurry were calculated and costs determined. These facts are shown in Table V.

Information on slurry composition and compressive strength is shown in Table V for four slurries which meet the specifications and would be suitable for use in filling mine drifts between bulkheads. From these four slurries, Grout Slurry No. 7 was selected because of its high compressive strength and low cost, and because it was more dense, less permeable and required less blending than the other slurries. This slurry, containing 56 parts Cement, 44 parts Fly Ash, 9 parts Salt and 50 parts water by weight, costs $19.21 per cubic yard.

The application of using a grout slurry to build a bulkhead remotely will be discussed in Section D of this report.

Information on bond of grout slurry to coal is given in Section B (3) of this report.

TABLE V

SLURRIES FOR FILLING MINE DRIFTS

Slurry No.	Materials per Cubic Yard of Slurry: Pounds: Cement	Fly Ash	Bentonite	Salt	Gallons Water	Slurry Weight Lbs./Cu.Yd.	Cost/Cu.Yd.
7	1060	832	--	170	114	3012	$19.21
8	907	714	32	184	123	2862	17.20
9	872	687	31	363	118	2916	18.33
10 *	325	325	--	---	136	2423	7.22

Slurry No.	Slurry Description**
7	56 parts Cement, 44 parts Fly Ash, 9 parts Salt; 50 parts water
8	56 parts Cement, 44 parts Fly Ash, 2 parts Bentonite + 11.5 parts Salt; 63 parts water
9	56 parts Cement, 44 parts Fly Ash, 2 parts Bentonite, 23 parts Salt; 63 parts water
10	25 parts Cement, 25 parts Fly Ash, 40 parts Gob; 87.5 parts water

* Contains 650 pounds gob

** Parts given are parts by weight

TABLE V (Continued)

SLURRIES FOR FILLING MINE DRIFTS

Slurry No.	Set Time Hours	Compressive Strength - Psi 24 Hrs.	48 Hrs.	72 Hrs.	7 Days
7	8	275	875	---	---
8	8	383	655	885	---
9	8	103	405	555	---
10 *	8	18	29	55	80

Slurry No.	Slurry Description**
7	56 parts Cement, 44 parts Fly Ash, 9 parts Salt; 50 parts water
8	56 parts Cement, 44 parts Fly Ash, 2 parts Bentonite + 11.5 parts Salt; 63 parts water
9	56 parts Cement, 44 parts Fly Ash, 2 parts Bentonite, 23 parts Salt; 63 parts water
10	25 parts Cement, 25 parts Fly Ash, 40 parts Gob; 87.5 parts water

* Contains 650 pounds gob

** Parts given are parts by weight

B. Expendable Grout Retainer Improvement

An Expendable Grout Retainer is a cloth container of sufficient size so that it can be placed in a mine drift opening, filled with grout slurry and allowed to obtain a set while conforming to the shape of the opening.

The objective of this phase of the tests was to evaluate the retainer and seek means of improving it. The work was accomplished in four parts, each of which will be discussed separately.

(1) Fabric Investigation: This part of the study was to determine the most desirable material with respect to size, permeability and strength which would effect a cost savings and increase efficiency.

It was thought that it might be advantageous to have the fabric used in the retainer construction to be of a permeable nature to permit a portion of the grout slurry to pass through the fabric as an aid in the sealing of the retainer to the floor and/or walls of the mine opening. A tentative wet strength of the fabric of 270 pounds per inch of width was established as a goal for the strength of the material.

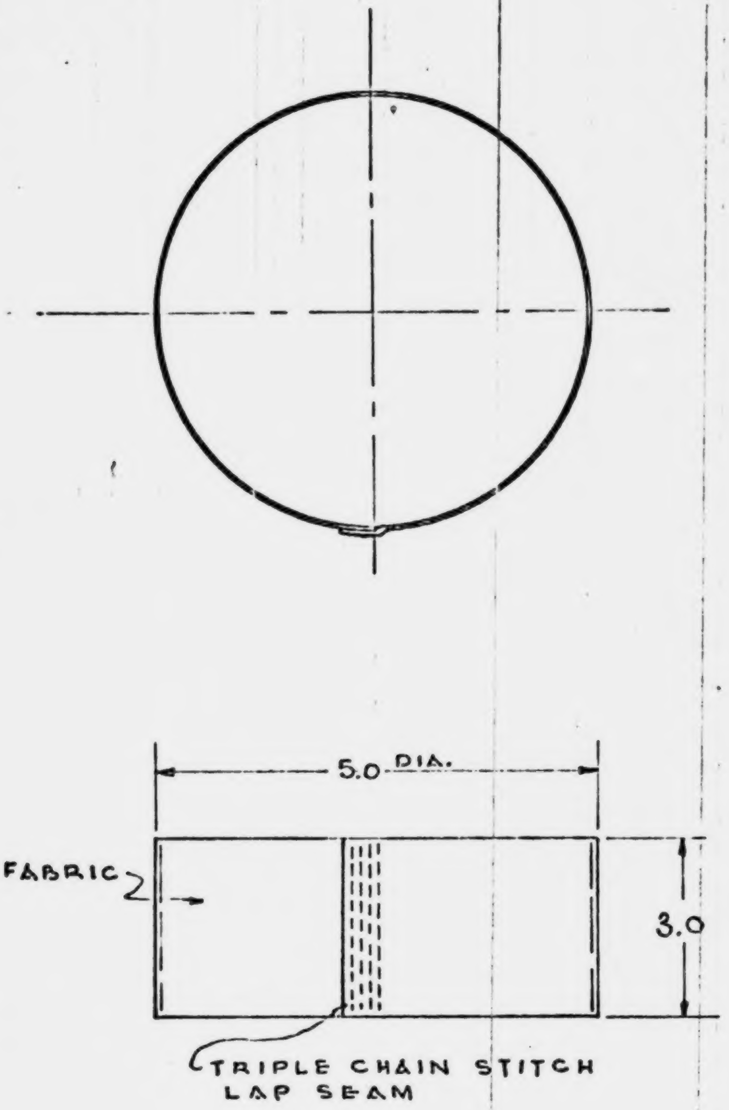
The first step in the test for the improvement of the Expendable Grout Retainer was to investigate fabrics which could be used in the construction of the retainer and which would meet the desired requirements for an improved retainer. Several different types of fabrics were obtained which were believed might meet the requirements as set forth above. These fabrics included cotton, polyethylene, nylon and dacron materials. Tensile test samples were made as shown in Figures 1 and 2. The samples were cut from the fabric in directions both parallel and perpendicular to the material selvage.

The test samples made as per Figure 1 used a triple chain stitch lap seam. The samples constructed in accordance with Figure 2 used a flat folded and double stitched type seam.

A total of 76 individual tests were made using 6 different fabrics. Tests were conducted on a Baldwin-Southwark tensile testing machine. This machine is shown in Figure 3 with a test specimen loaded in the machine. A closer view of a test specimen in the machine is shown in Figure 4. Prior to loading the test specimen in the machine, each seam area was soaked in water for one hour.

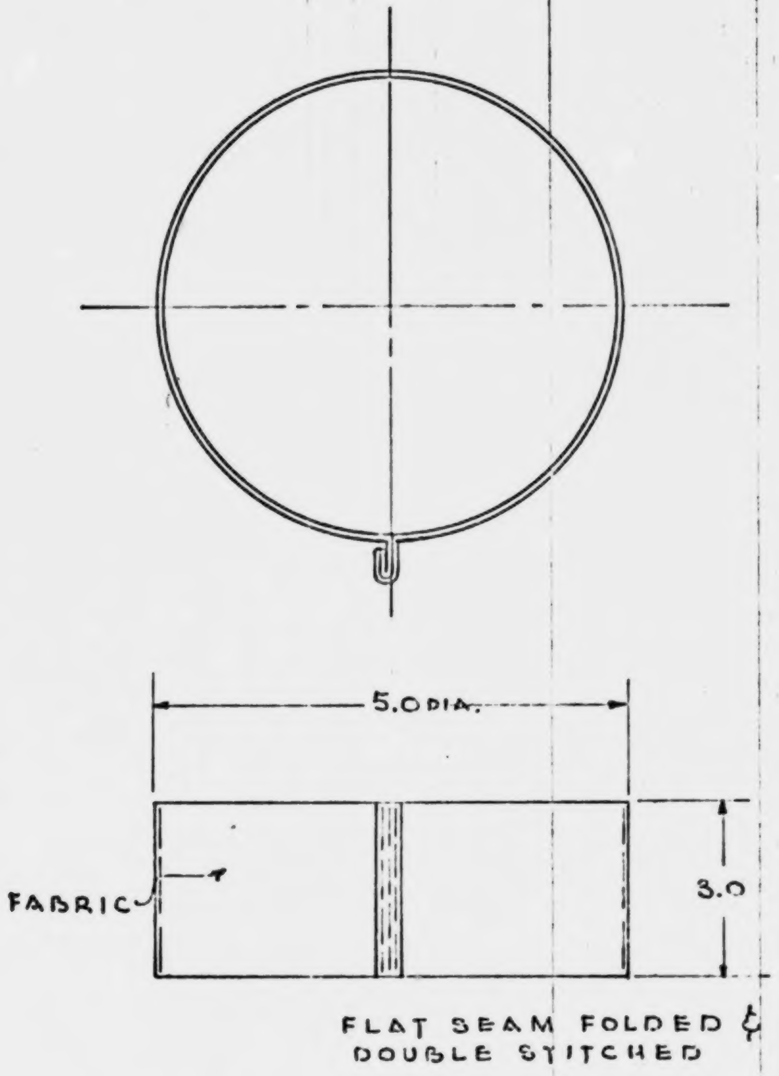
Test results are shown in Table VI. The table indicates the fabric, direction of cut of the fabric, type of seam, failure force, tensile strength and area of failure.

It will be noted that 36 of the tests were conducted on cotton fabrics and 40 of the tests were run on fabrics of other types of material. In the 76 tests, 35 of the test specimens failed at the seams. Thirteen of these were cotton fabrics and 22 were other materials. In the 35 seam failures, 28 were of the flat folded and double stitched type seam and only 7, or 20%, were of the triple chain stitch lap seam.


5.0 DIA.
FABRIC
3.0
TRIPLE CHAIN STITCH
LAP SEAM

FABRIC TENSILE TEST SAMPLE
TRIPLE CHAIN STITCH LAP SEAM

FIGURE 1


5.0 DIA.
FABRIC
3.0
FLAT SEAM FOLDED &
DOUBLE STITCHED

FABRIC TENSILE TEST SAMPLE
FLAT FOLDED DOUBLE STITCHED SEAM

FIGURE 2

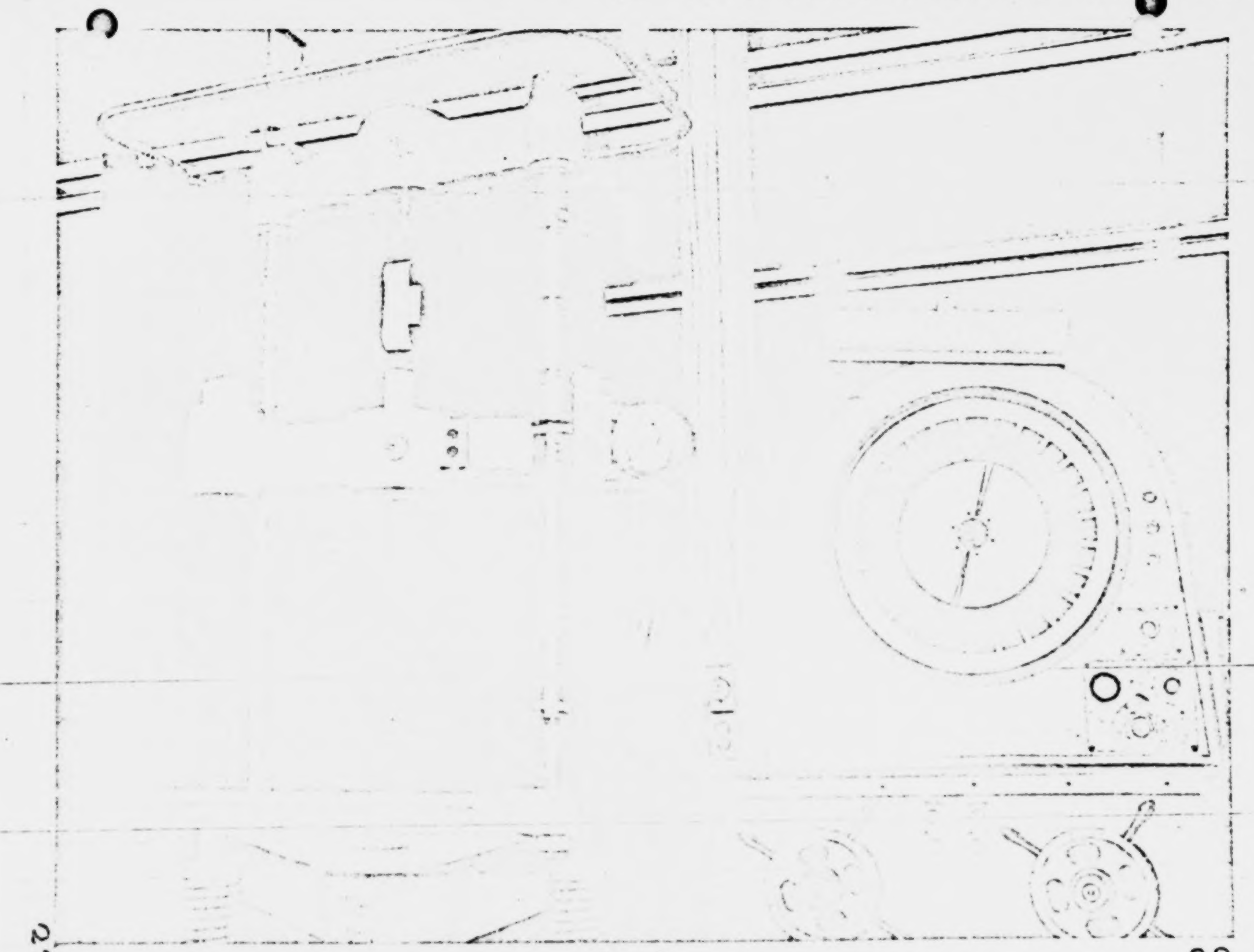


FIG. 3

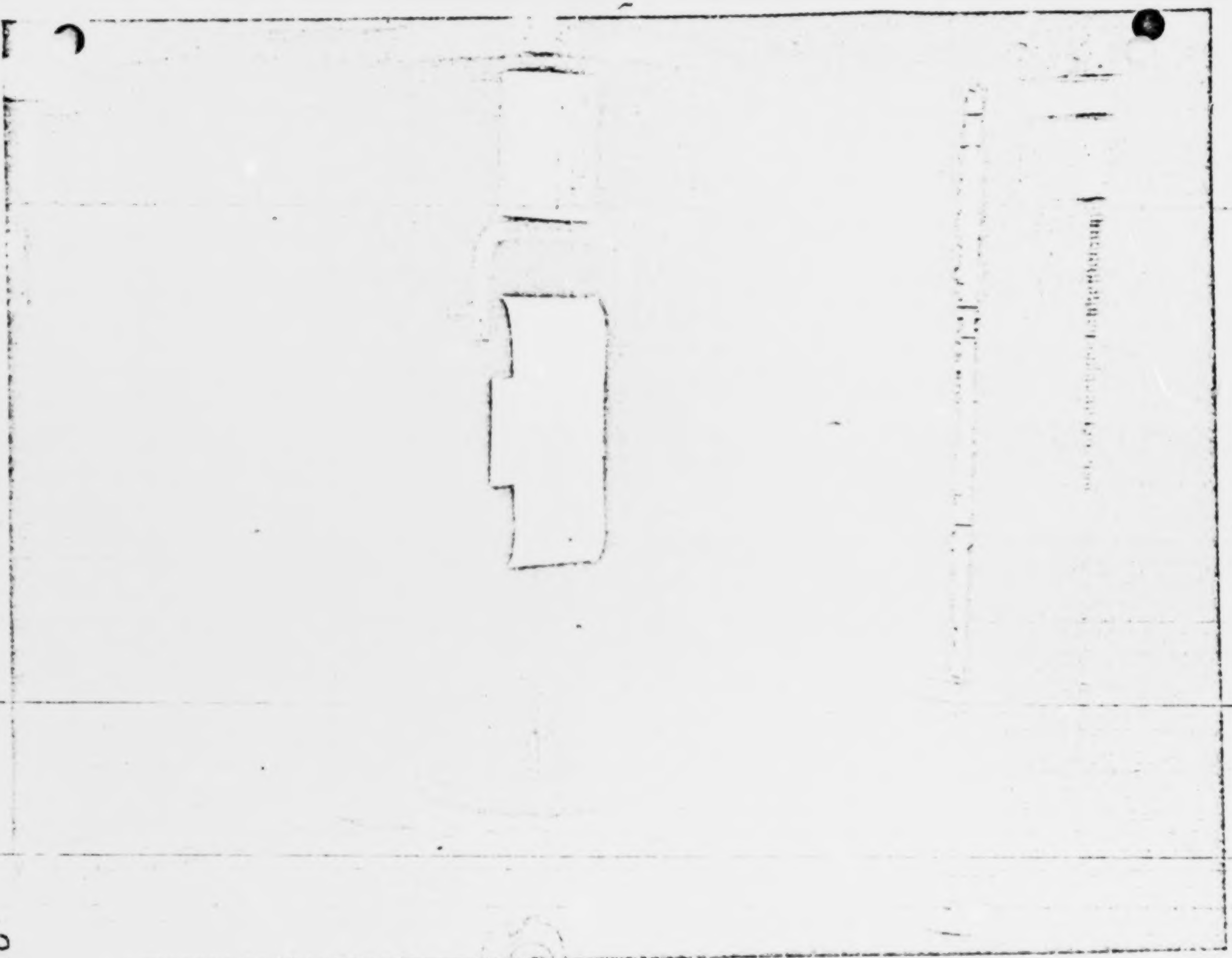


FIG. 4

TABLE VI

FABRIC TENSILE TEST RESULTS

Fabric*	Cut of Fabric In Relation to Selvage Parallel or Perpendicular	Seam Type**	Failure Force Lbs.	Tensile Strength Lbs. Per In. Of Width*** Avg.	Max.	Failure Area
A	Parallel	Flat	600			Seam
A	Parallel	Flat	545	142	150	Fabric
A	Parallel	Flat	565			Seam
A	Parallel	Lap	515			Fabric
A	Parallel	Lap	530	129	132	Fabric
A	Parallel	Lap	510			Fabric
A	Perpendicular	Flat	375			Fabric
A	Perpendicular	Flat	375	93	94	Fabric
A	Perpendicular	Flat	365			Fabric
A	Perpendicular	Lap	365			Fabric
A	Perpendicular	Lap	350	87	91	Fabric
A	Perpendicular	Lap	335			Fabric
B	Parallel	Flat	365			Fabric
B	Parallel	Flat	350	89	91	Seam
B	Parallel	Flat	360			Seam
B	Parallel	Lap	345			Fabric
B	Parallel	Lap	340	86	87	Seam
B	Parallel	Lap	350			Fabric
B	Perpendicular	Flat	290			Seam
B	Perpendicular	Flat	250	67	72	Fabric
B	Perpendicular	Flat	260			Fabric
B	Perpendicular	Lap	245			Fabric
B	Perpendicular	Lap	235	61	64	Fabric
B	Perpendicular	Lap	255			Fabric
C	Perpendicular	Lap	625			Fabric
C	Perpendicular	Lap	635	152	159	Fabric
C	Perpendicular	Lap	560			Fabric
C	Parallel	Lap	615			Fabric
C	Parallel	Lap	595	147	154	Fabric
C	Parallel	Lap	555			Fabric

TABLE VI (Continued)

FABRIC TENSILE TEST RESULTS

Fabric*	Cut of Fabric In Relation to Selvage Parallel or Perpendicular	Seam Type**	Failure Force Lbs.	Tensile Strength Lbs. Per In. Of Width***		Failure Area
				Avg.	Max.	
C	Parallel	Flat	540			Seam
C	Parallel	Flat	485	126	135	Seam
C	Parallel	Flat	490			Seam
C	Perpendicular	Flat	475			Seam
C	Perpendicular	Flat	480	119	120	Seam
C	Perpendicular	Flat	480			Seam
D	Parallel	Lap	330			Fabric
D	Parallel	Lap	340	84	85	Fabric
D	Parallel	Lap	340			Fabric
D	Parallel	Flat	340			Seam
D	Parallel	Flat	275	73	85	Seam
D	Parallel	Flat	265			Seam
D	Perpendicular	Flat	130			Seam
D	Perpendicular	Flat	135	34	36	Seam
D	Perpendicular	Flat	130			Fabric
D	Perpendicular	Flat	145			Fabric
D	Perpendicular	Lap	75			Seam
D	Perpendicular	Lap	85	19	21	Seam
D	Perpendicular	Lap	70			Seam
E	Perpendicular	Lap	705			Fabric
E	Perpendicular	Lap	590	159	176	Fabric
E	Perpendicular	Lap	615			Fabric
E	Parallel	Lap	435			Seam
E	Parallel	Lap	385	99	109	Fabric
E	Parallel	Lap	375			Fabric
E	Perpendicular	Flat	430			Seam
E	Perpendicular	Flat	360	99	107	Seam
E	Perpendicular	Flat	395			Seam

TABLE VI (Continued)

FABRIC TENSILE TEST RESULTS

Fabric*	Cut of Fabric In Relation to Selvage Parallel or Perpendicular	Seam Type**	Failure Force Lbs.	Tensile Strength Lbs. Per In. Of Width *** Avg.	Max.	Failure Area
E	Parallel	Flat	325			Seam
E	Parallel	Flat	310	82	87	Seam
F	Parallel	Flat	350			Seam
F	Parallel	Lap	365			Fabric
F	Parallel	Lap	345	88	91	Fabric
F	Parallel	Lap	350			Fabric
F	Perpendicular	Lap	260			Seam
F	Perpendicular	Lap	265	64	66	Fabric
F	Perpendicular	Lap	240			Seam
F	Perpendicular	Flat	245			Seam
F	Perpendicular	Flat	250	62	62	Seam
F	Perpendicular	Flat	245			Seam
F	Parallel	Flat	215			Seam
F	Parallel	Flat	250	60	64	Seam
F	Parallel	Flat	255			Seam

* Fabric Description

Fabric No.	Type of Fabric	Weave	Thd. Ct.	Weight (Oz./Sq.Yd.)	Cost Per Sq. Yd.
A	Cotton	Plain	26 x 22	10.5	$ 0.75
B	Sgl. Fill Cotton	Oxford	44 x 20	7.5	0.59
C	Spun Nylon	Plain	36 x 38	6.5	2.41
D	Polyethylene Mesh	Plain	30 x 18	2.0	1.70
E	Cotton	Plain	26 x 20	10.0	0.75
F	Spun Dynel	Oxford	34 x 36	5.5	2.46

** Lap - Triple Chain Stitch Lap Seam
Flat - Flat Folded and Double Stitched Type Seam

*** Strength figures based on average fabric width of 2 inches.

On the basis of these results, the triple chain stitch lap seam proves to be the most desirable seam to be used in the construction of the Expendable Grout Retainer and the cotton material retains the seam better than the other materials tested.

Table VI shows that Fabric "E" exhibited the highest strength of any fabric sample tested, but was not consistently as strong as Fabric "C". However, the strength of Fabric "E" is believed to be sufficient and the cost is substantially less, so Fabric "E" was selected based primarily on the cost factors as reflected in the data of Table VI. Fabric "E" is a 10-ounce cotton material with a 20 x 26 thread count. It has a maximum tensile strength of 176 and 109 pounds per inch of width, considering cuts of material both perpendicular and parallel to selvage. The cost is $0.75 per square yard.

However, several factors related to these tests should be pointed out in relation to the results obtained. It is believed that the results obtained in these tests are conservative values because the test was made on a small sample of the material. It is felt that a continuous seam in a large fluid filled retainer would exhibit a greater strength per inch than that obtained by the seams in the narrow sample used in these tests, due to the discontinuity necessarily present at each end of the sample seam. It is also feasible that a large retainer would be subjected to a more uniform load than could be applied with a testing machine.

Therefore, the maximum fabric strength of 176 pounds per inch obtained in the tests is believed sufficient, even though it is only 65% of the tentative strength of 270 pounds per inch originally selected.

Because of the preceding reasons, it is felt that the strength of 270 pounds per inch tentatively established for the seam is excessive. It was also noted in the tests that severe distortion of the threads in the fabrics occurred adjacent to the seam prior to maximum load which caused failure. The gaps caused by this distortion could be a limiting factor on the amount of grout which the retainer could hold and would provide an indication of imminent retainer failure. Since a variety of materials had been tested, tensile tests were concluded.

Four of the fabrics were used to construct small receptacles for laboratory tests to determine the amount of slurry loss through the material. Tests were made with the fabric both dry and wet. These fabrics are listed below:

Fabric No.	Type of Fabric	Weave	Thd. Ct.	Weight (Oz./Sq.Yd.)
B (#2)	Sgl. Fill Cotton	Oxford	44 x 20	7.5
C (#3)	Spun Nylon	Plain	36 x 38	6.5
D (#4)	Polyethylene Mesh	Plain	30 x 18	2.0
F (#5)	Spun Dynel	Oxford	34 x 36	5.5

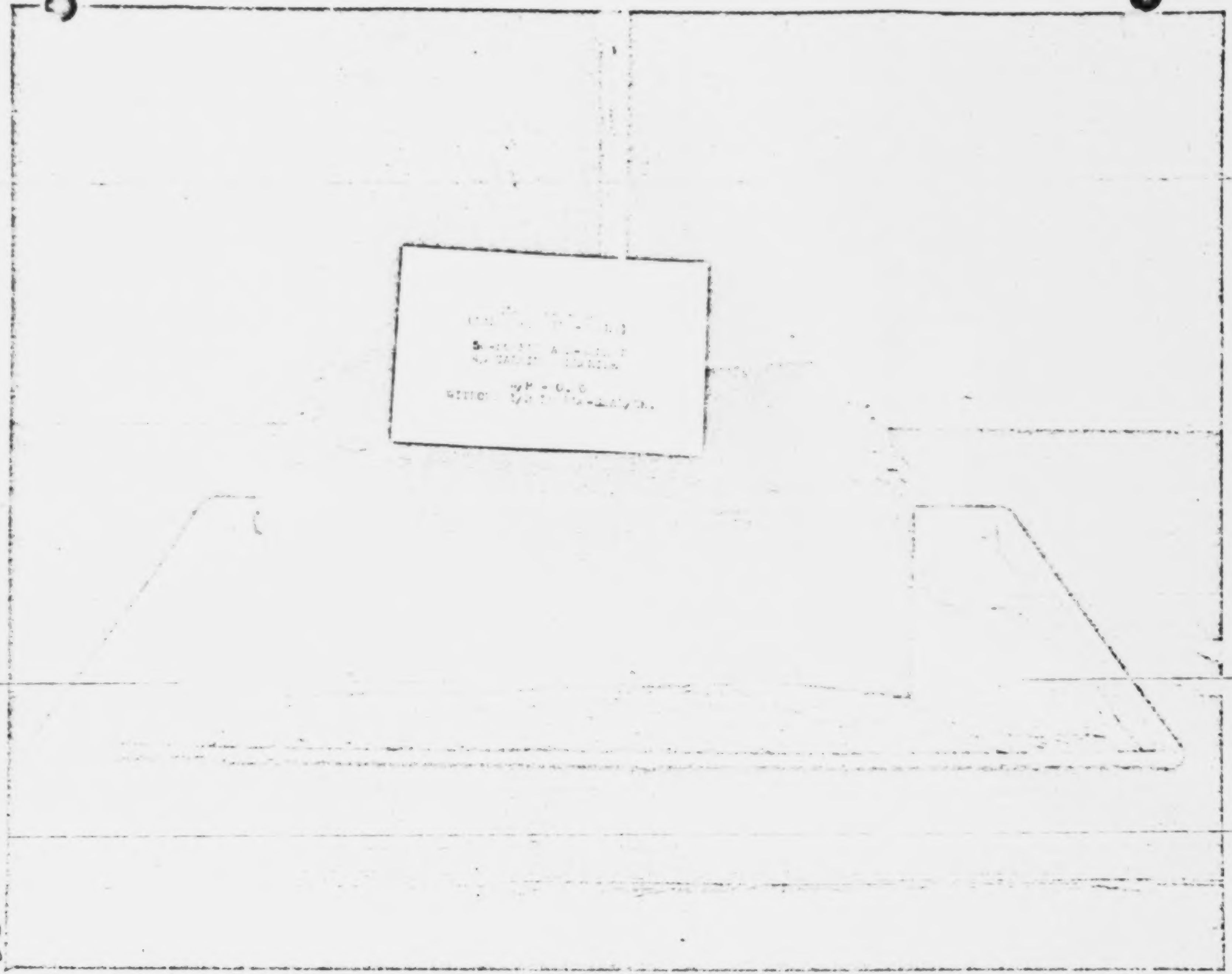
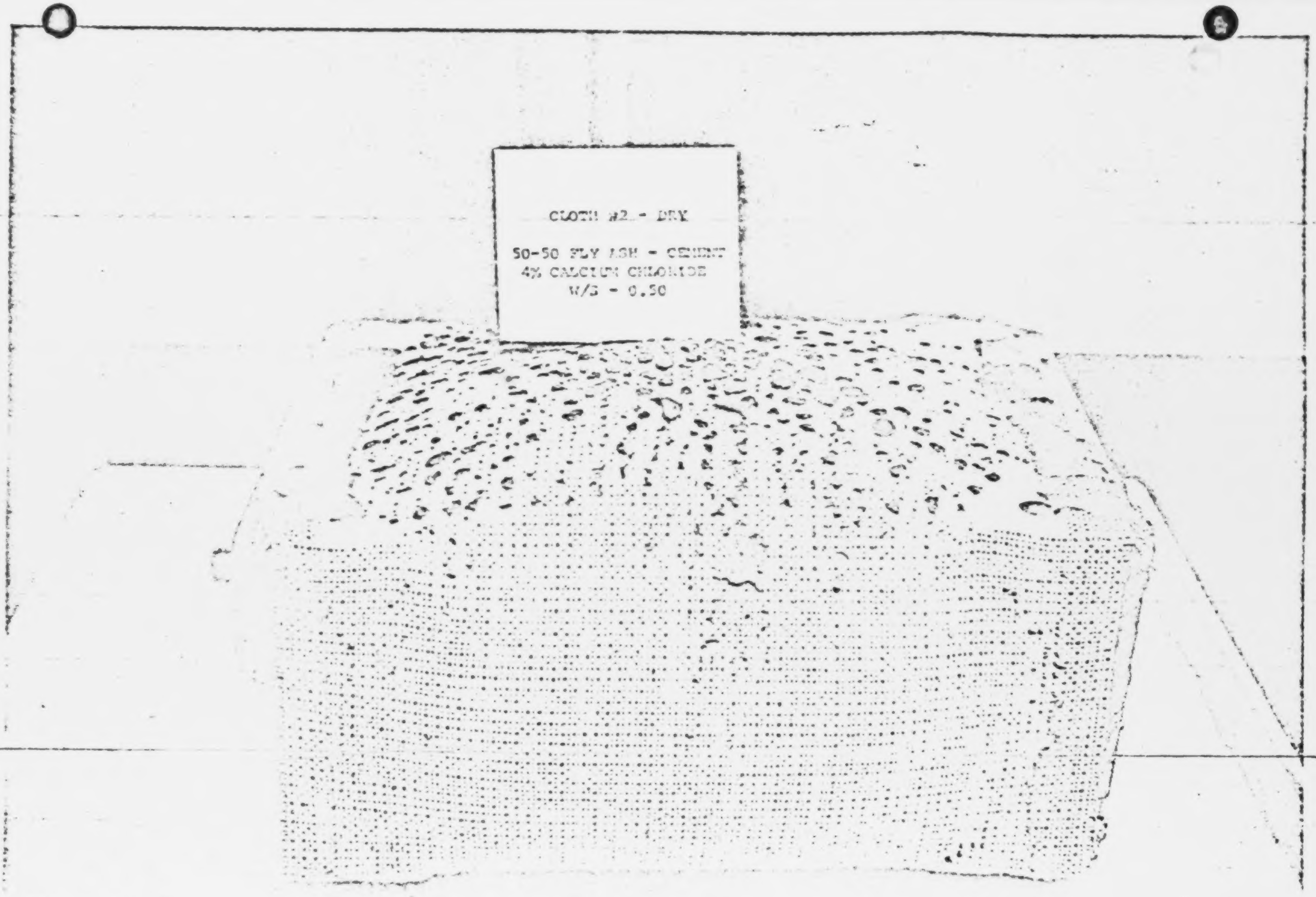


FIG. 5

Reproduced from best available copy.


CLOTH #2 - DRY
50-50 FLY ASH - CEMENT
4% CALCIUM CHLORIDE
W/S - 0.50

[illegible]

Fabric No. 4 was a polyethylene mesh. The openings in the fabric were too large and allowed the slurry to flow out of the receptacle as fast as it was poured in. A photograph of this test is shown in Figure 5.

Fabrics 2, 3 and 5 allowed only a small amount of slurry to pass and then only water would escape through the permeability of the cloth. A photograph of a typical test using this material is shown in Figure 6.

The grout slurry used in these tests was Grout Slurry No. 2 composed of 56 parts Cement, 44 parts Fly Ash, 4 parts Calcium Chloride and 50 parts water by weight. The slurry was poured into the samples through a hose with a head of about two feet on the slurry.

Based on these test results, it was concluded that it would be very difficult to control the slurry loss with the fabric permeability. Therefore, it was considered impractical to use a permeable fabric for the purpose of passing slurry to the outside of the retainer. The area around the retainer would be sealed by coating the fabric and surrounding area with the grout slurry just prior to filling the Expendable Grout Retainer with the grout slurry.

As a result of the series of tests, the 10-ounce cotton Fabric "E" having a 20 x 26 thread count and a tensile strength of 176 pounds per inch of width was selected as the material for construction of Expendable Grout Retainers and Auger Hole Plugs.

(2) Small-scale Tests: In this part of the investigation, the purpose was to establish the burst pressure and characteristics of small-scale retainers simulating the Expendable Grout Retainers. The approximate burst pressure was calculated using a modified version of the formula for hoop stress in a thin-walled steel cylinder. This formula is:

$$P = \frac{S\,t}{r}$$

where

P = Internal pressure, psi

S = Tensile stress, psi

t = Thickness of cylinder wall, inches

r = Radius of cylinder

For the purposes of this test, the formula was modified to eliminate the thickness factor for the cloth containers. This formula then became:

$$P = \frac{S}{r}$$

or, by using diameter, D,

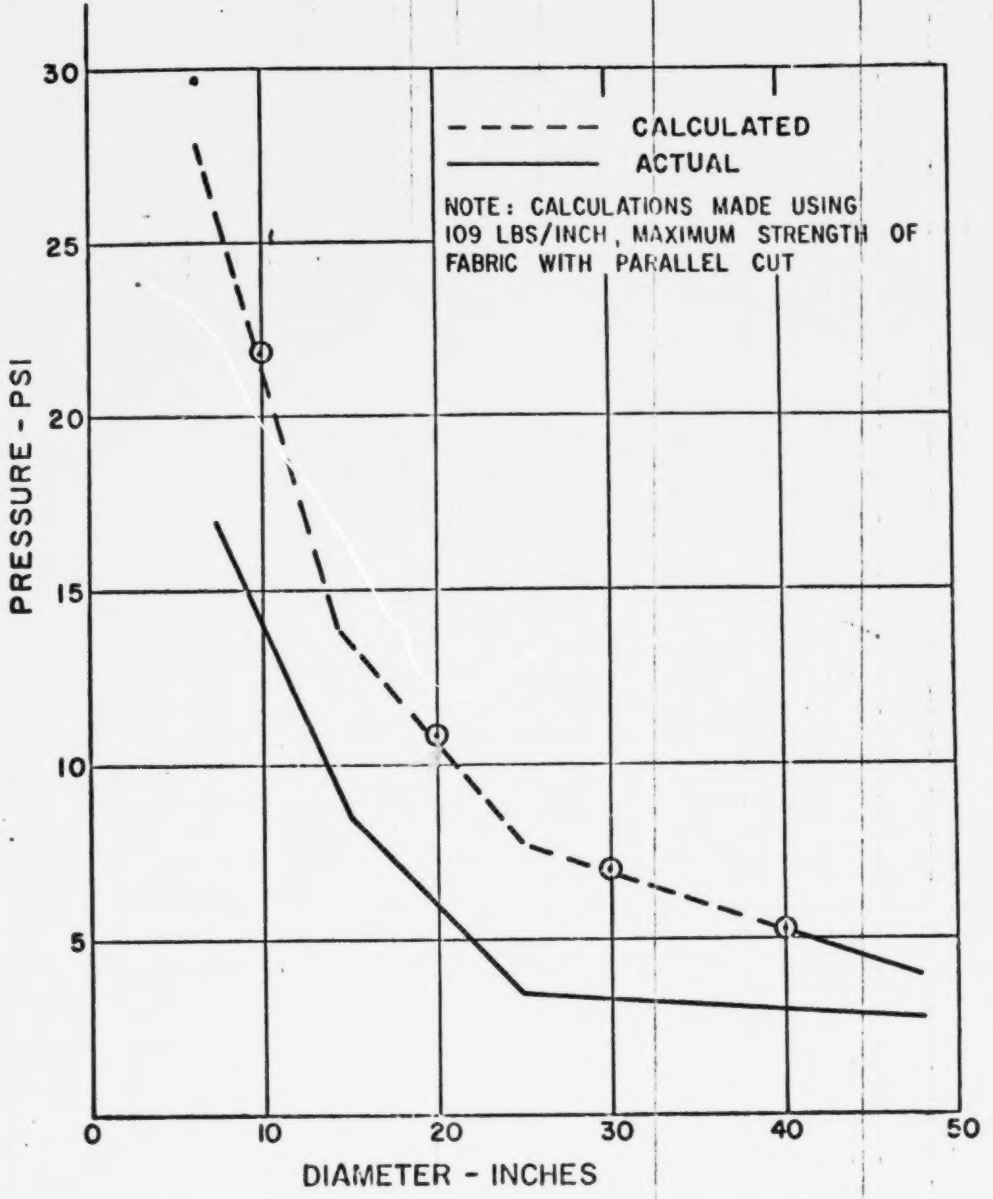
$$P = \frac{2\,S}{D}$$

where S = Tensile strength of fabric, pounds per inch of width

The selected material to be used in building the retainers had a maximum tensile strength of 176 pounds per inch of width in a direction perpendicular to the selvage, but only 109 pounds per inch of width in the direction of stress in the scale model, because in the construction of the model the cut of the material was in a direction parallel to the selvage. The selected triple chain stitch lap seam was used in the construction. The anticipated burst pressure for a range of diameters was calculated from the above mentioned formula and the results are shown in the graph attached as Figure 7.

It should be noted that the anticipated burst pressure calculated from the formula is about twice that actually obtained. This could be due to the fact that the formula basically applies to a steel cylinder with a known thickness of uniform strength, whereas the fabric is not of uniform consistency and individual threads can effect the overall strength. However, the actual burst pressure varies in relation to the diameter in the same proportion as it does in the value calculated from the formula, so the formula could be used to obtain an approximate burst pressure by dividing the result obtained by two or by simply changing the formula from $P = \frac{2S}{D}$ to $P = \frac{S}{D}$. Then an appropriate safety factor could be inserted for actual field usage.

Nine burst tests were made on the small-scale retainers. Tests were conducted by pumping a cement or gelled water slurry (shown in Table VII) with a centrifugal pump at approximately 20 gallons per minute through a 1-inch hose into the cloth retainer until failure occurred. Flow was controlled by manual manipulation



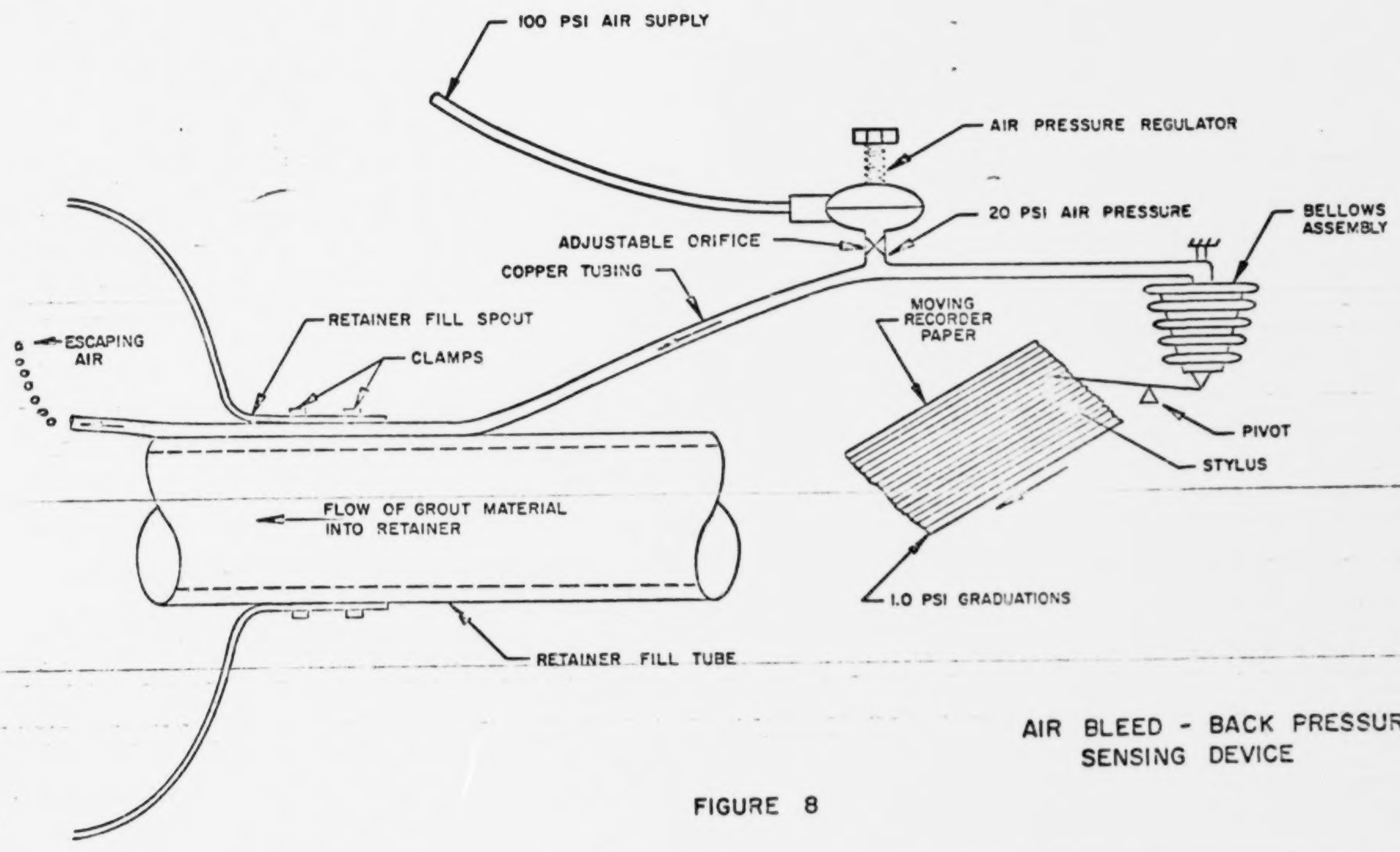
BURST PRESSURE
SMALL SCALE RETAINER TESTS

FIGURE 7

of a valve in the flow line. Pressure was recorded using an air bleed-back pressure sensing device as shown in Figure 8. Initial tests were conducted on retainers having flat dimensions of 20 inches in width and 48 inches in length. Figure 9 shows this retainer just prior to initiation of filling with slurry. After filling, the retainer expanded to a mean diameter of 15 inches. Some slurry extruded through the weave of the fabric. This is shown in Figure 10. Three tests were made on this size retainer in an unrestricted condition. One retainer failed due to faulty construction around the spout attachment. The remaining two failed at 8 and 9 psi. Three other retainers of this size were tested by placing in a fixture which restricted the vertical expansion to 7.5 inches. This test layout is shown in Figure 11. The burst pressure obtained in the tests was 17 psi. It can be noted that the place of failure on these retainers is on the bottom side in the area restricted by the test fixture instead of the open side area where the failure might normally be expected. The only logical explanation is that failure will occur at the weakest point, so the place of failure may vary as the construction varies on the fabric used.

Two other tests were conducted on larger size retainers having diameters of 25 and 48 inches in an unrestricted condition. These retainers burst at pressures of 3.48 psi and 2.58 psi respectively. The larger size was also tested inside a steel

1. AIR ALLOWED TO ESCAPE INTO RETAINER THROUGH COPPER TUBING.
2. THE PRESSURE REQUIRED TO CAUSE THE AIR TO ESCAPE IS IMPOSED ON BELLOWS ASSEMBLY.
3. MOVEMENT OF BELLOWS ASSEMBLY PROPORTIONAL TO IMPOSED PRESSURE.
4. THIS MOVEMENT RECORDED ON RECORDER CHART BY STYLUS.


100 PSI AIR SUPPLY
AIR PRESSURE REGULATOR
20 PSI AIR PRESSURE
BELLOWS ASSEMBLY
ADJUSTABLE ORIFICE
COPPER TUBING
RETAINER FILL SPOUT
ESCAPING AIR
CLAMPS
MOVING RECORDER PAPER
PIVOT
STYLUS
FLOW OF GROUT MATERIAL INTO RETAINER
1.0 PSI GRADUATIONS
RETAINER FILL TUBE

AIR BLEED - BACK PRESSURE
SENSING DEVICE

FIGURE 8

FIG. 9

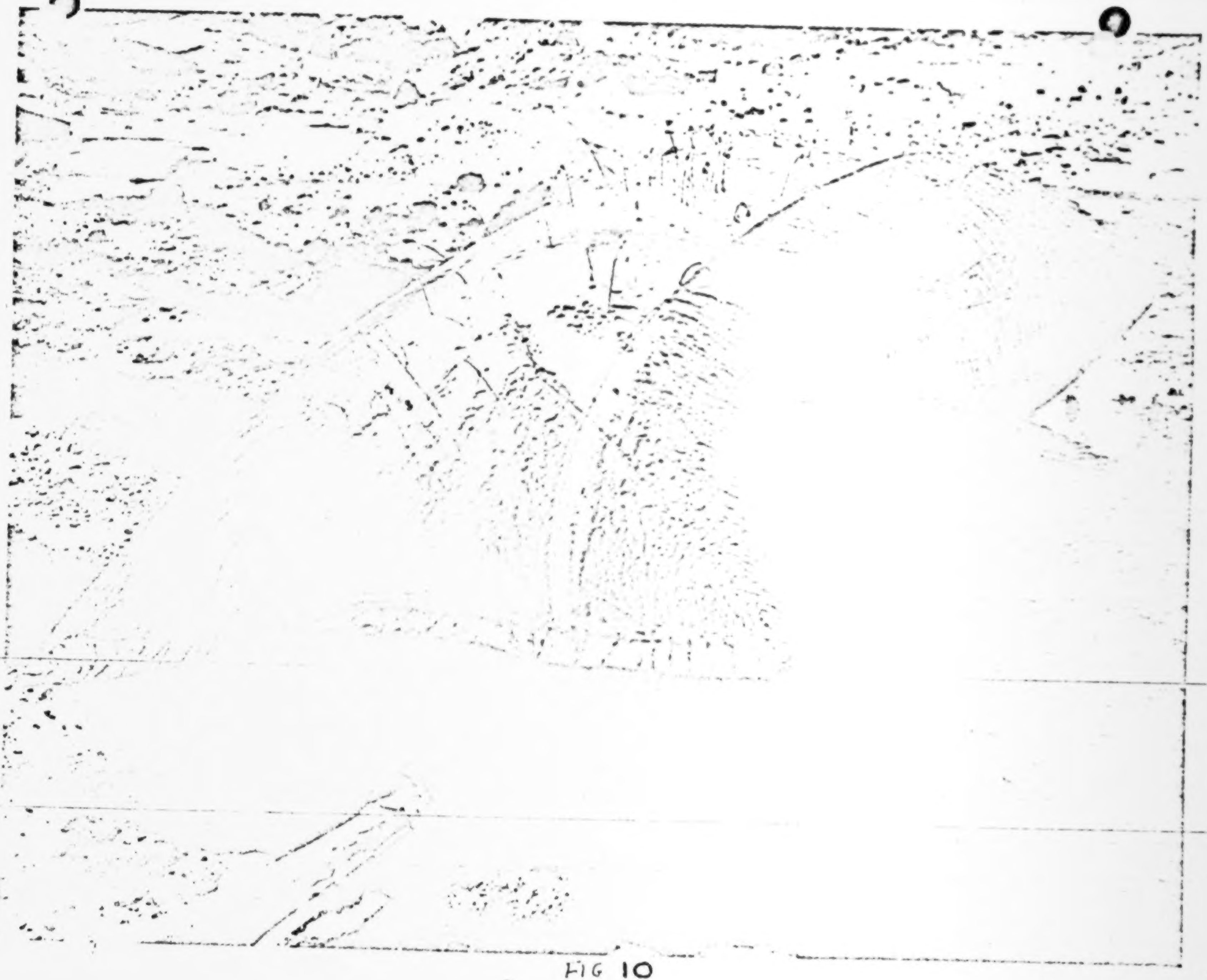


FIG 10

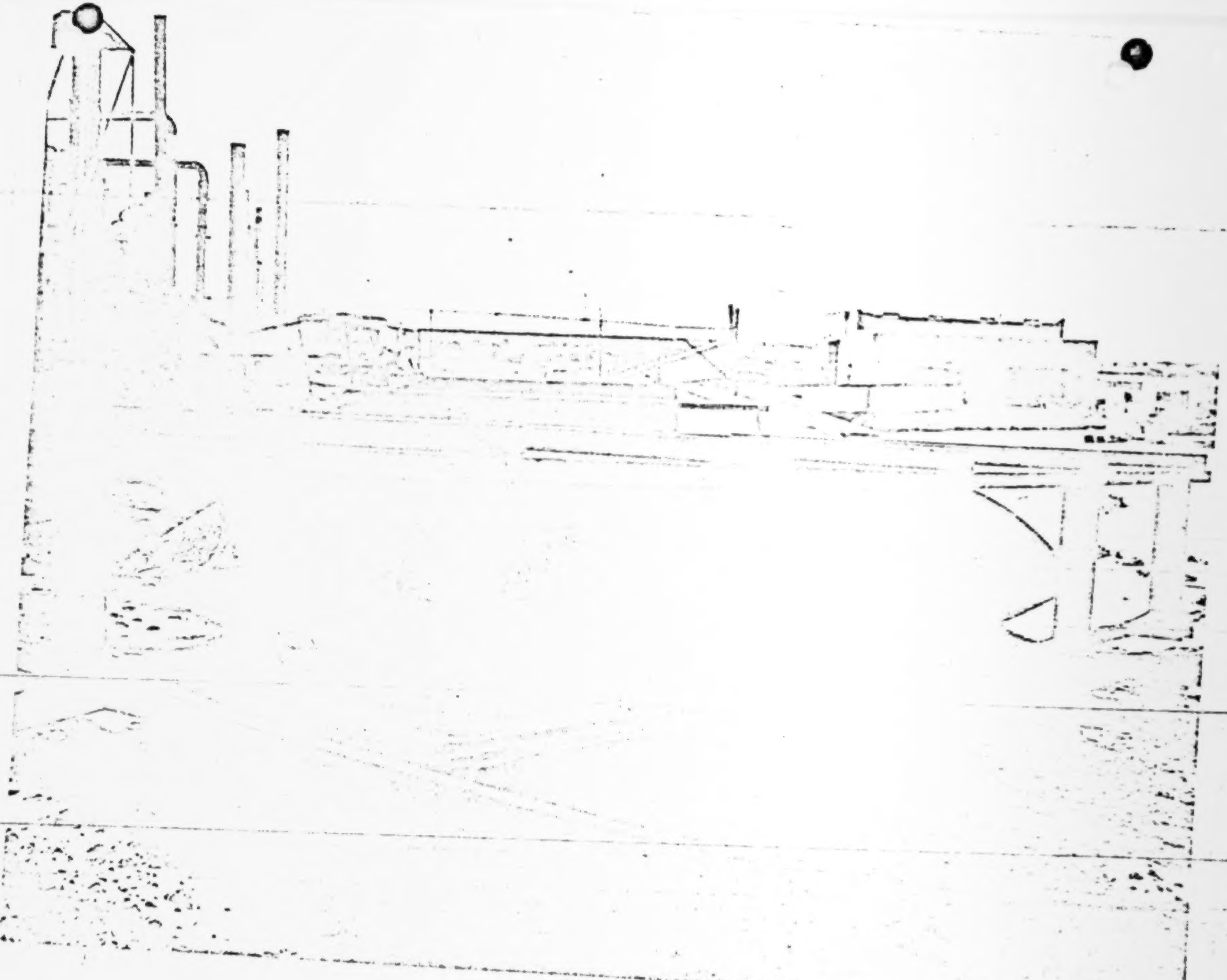


FIG. 11

casing, but data obtained was incomplete because the pressure exceeded the range of the recording equipment. The retainer was restrained on the open end of the casing by a chain used to hold the steel fill pipe in place. Test data is shown in Table VII.

An investigation was also made to confirm the feasibility of placing an aggregate fill in a container and subsequently grouting in place.

Laboratory tests were made using small cloth bags filled with various grades of aggregate. A tube was inserted to half the depth of the gravel and grout slurry was introduced until the grout stood 2 feet high in the tube. The grout was allowed to harden and then the specimens were sawed in half for observation. Figure 12 shows the test set-up and the results of the bag containing graded 3/4-inch gravel. Figure 13 shows the specimens containing graded 3/8-inch gravel and ungraded tailings (1/2-inch to dust). It can be noted that the graded aggregate specimens were completely grouted, but the one containing the fines was only grouted in the sides and upper area, and the center bottom portion was still unconsolidated. Another test was made using the ungraded aggregate with the tube only one-half inch from bottom, but results were duplicated. Slurry used was Slurry No. 2, composed of 56 parts Cement, 44 parts Fly Ash, 4 parts Calcium Chloride and 50 parts water by weight.

TABLE VII

SMALL-SCALE RETAINER BURST TESTS

Test No.	Retainer Dia.-Inches	Test Condition	Pressure Fluid *	Burst Pressure Psi	Calculated Burst Pressure Psi
1	15	Unrestricted	Cement Slurry	6 **	14.5
2	15	Unrestricted	Cement Slurry	8	14.5
3	15	Unrestricted	Cement Slurry	9	14.5
4	48	Unrestricted	Gelled Water	2.58	4.55
5	25	Unrestricted	Gelled Water	3.48	8.82
6	7.5	Restricted Top & Bottom	Cement Slurry	11.2 ***	29.0
7	7.5	Restricted Top & Bottom	Cement Slurry	17.0	29.0
8	7.5	Restricted Top & Bottom	Cement Slurry	17.0	29.0
9	49	In Steel Csg.	Gelled Water	---- ****	4.45

* Cement slurry contained 20% Cement, 40% Fly Ash, and 40% Gypsum mixed to a slurry weight of 13.8 lbs./gal. Gelled water contained Bentonite and Cellophane Flake mixed to slurry weight of 8.5 lbs./gal.

** Retainer had faulty construction in area of spout attachment and failed prematurely.

*** Retainer did not burst due to low capacity pumping equipment.

**** Pressure recorder had a maximum pressure capability of 12 psi. This pressure was exceeded and continued to rise until retainer burst. Based on results of Test No. 4 and calculated value for this test, anticipated pressure was less than 12 psi, so a visual gauge was not used in the test.

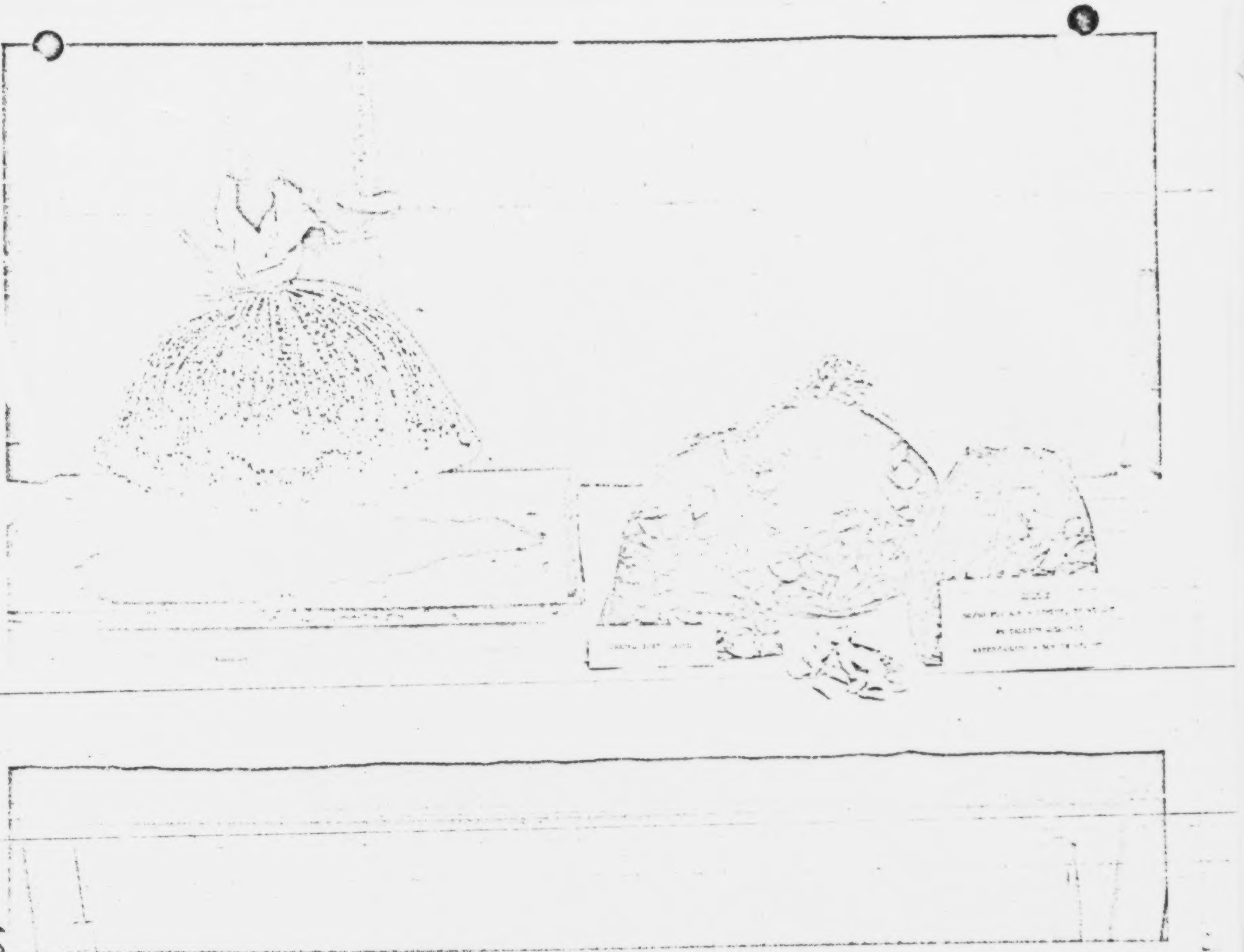


FIG. 12

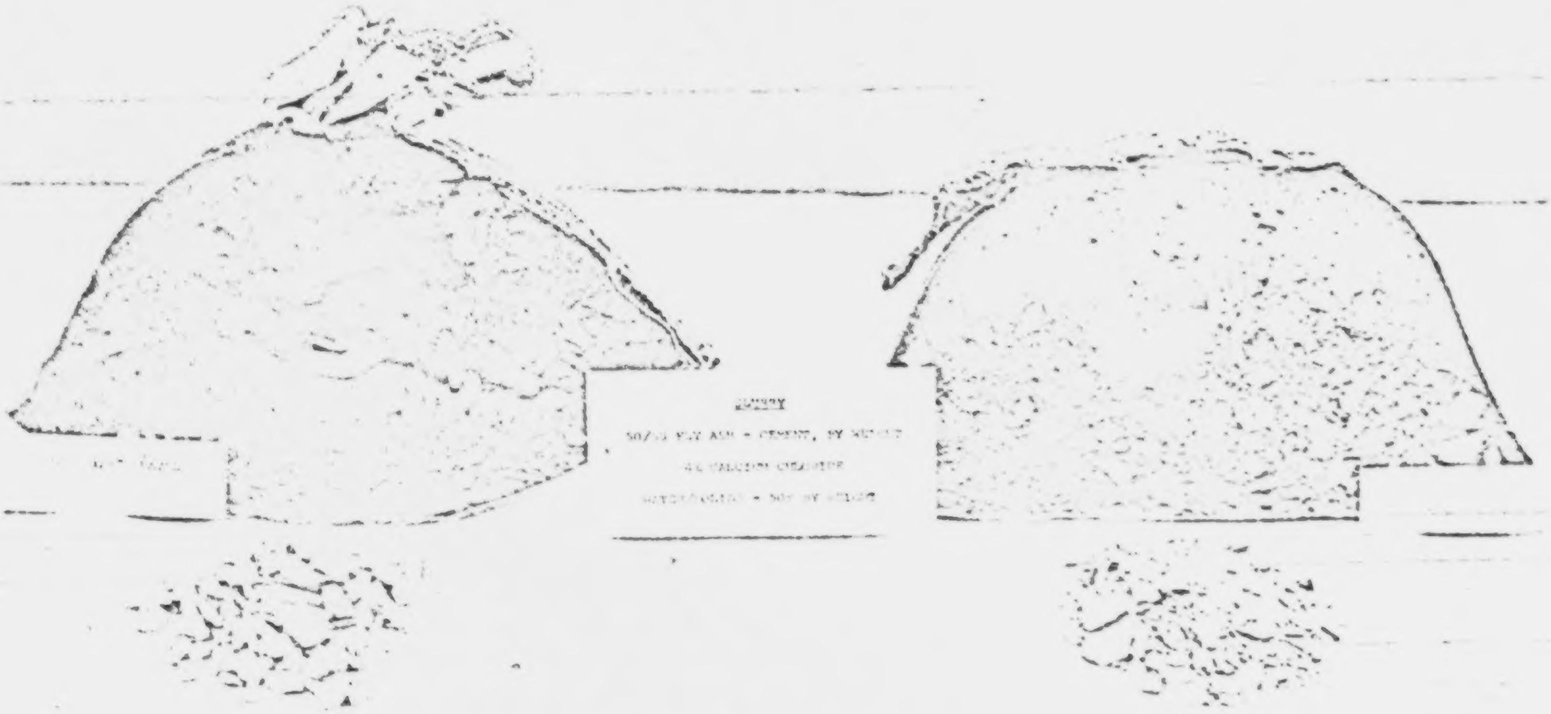

SLURRY
50/50 FLY ASH - CEMENT, BY WEIGHT
CALCIUM CHLORIDE

FIG. 13.

An additional test was made on a small-scale basis to evaluate the grouting of a bulkhead of aggregate using a quick-setting grout slurry. The test fixture was a wooden box constructed to enable complete disassembly without disturbing the grouted material.

The test box was filled with a bulkhead of crushed rock. The rock used for the test was a graded 3/8-inch gravel aggregate free of fine material. About 4-1/2 cubic feet of rock was contained in the bulkhead.

As the bulkhead was placed in the box, three 3/8-inch grout pipes 30 inches long were placed in a vertical line on 6-inch centers, with the lower pipe 3 inches from the bottom of the test box. The outlet ends were open and were located in the approximate center of the bulkhead.

The grout slurry selected from laboratory tests was a mixture of two slurries in equal volumes. The cement portion of the slurry contained 58 parts Type I Portland Cement, 42 parts Silica Flour and 72 parts water. The silicate portion of the slurry contained 52 parts Silica Flour, 48 parts Sodium Silicate and 44 parts water. Four gallons of each slurry were prepared. The two slurries were mixed together in equal volumes through a mixer head just prior to being pumped in the aggregate bulkhead. The mixed grout was pumped into the bottom grout pipe first at a rate of 3 gallons per minute. The eight gallons were pumped into the bulkhead and then repeated for the center and upper grout pipes. The fluid time for the grout slurry was 35 seconds after mixing the two portions together. Compressive strength samples of the grout slurry indicated a strength of 987 psi in 24 hours and 916 psi in 72 hours.

After 18 hours the box was disassembled and all loose rock was removed. The remaining portion of the grouted bulkhead is shown in Figure 14. The compressive strength of the grouted aggregate was 767 psi in 24 hours and 823 psi in 72 hours. The maximum movement of the grout slurry through the crushed rock was 51 inches from the discharge of the grout pipe.

Based on the amount of rock consolidated, it was concluded that the placement of grouting pipes was of utmost importance with the quick-set grout. A proper distribution across the upper row would have given complete consolidation of the bulkhead. It was also concluded that the grout distribution could be controlled to some extent by varying the set time.

The burst tests indicated (See Figure 7) that a pressure of 3 psi should not be exceeded in a full-scale retainer test where it is filled to 30 inches height. For the top retainer, a maximum pressure of 5 psi can be utilized since the height is less.

The tests also confirmed that the characteristics of a grout retainer were such that the shape conformed to its surroundings in a satisfactory manner.

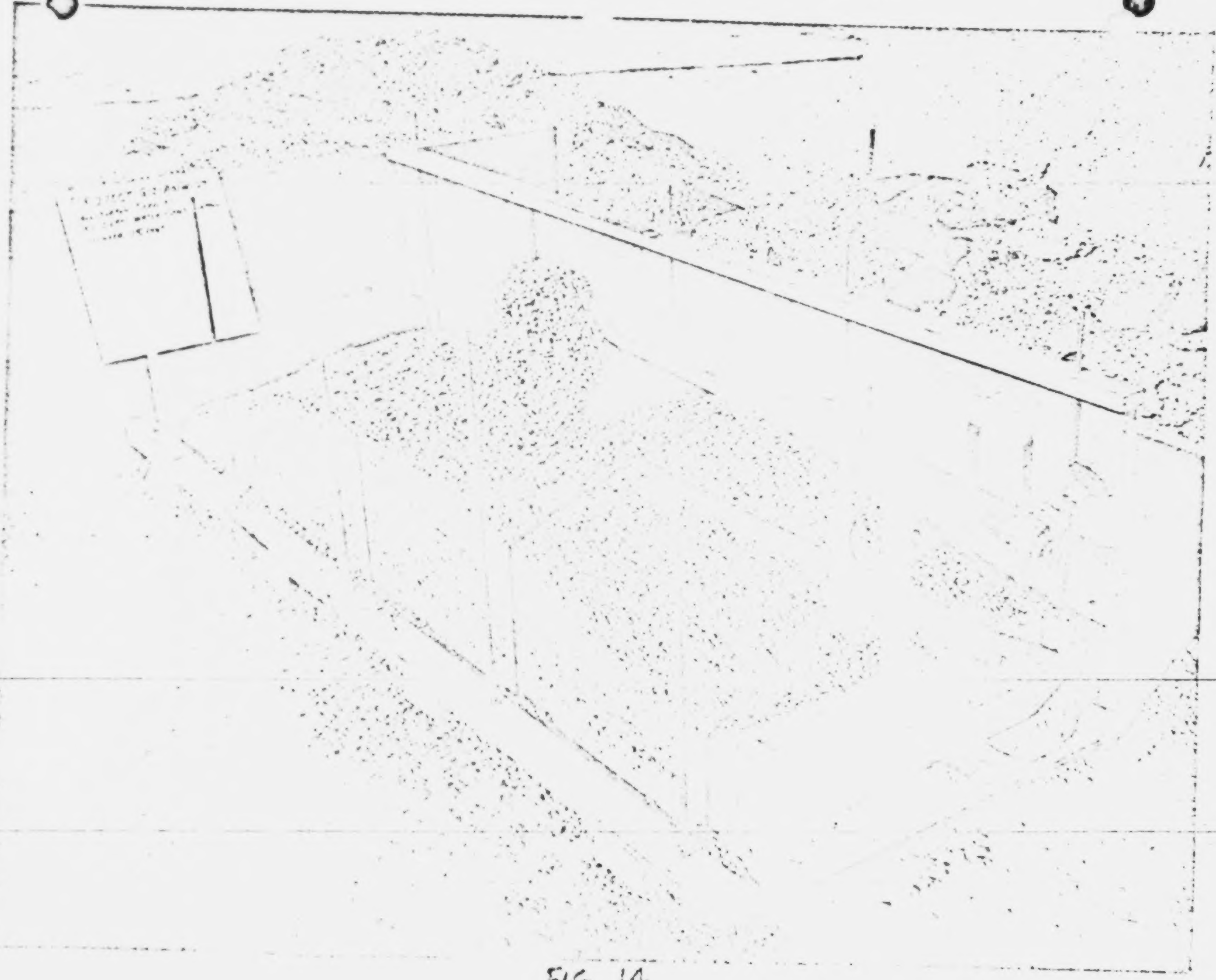


FIG. 14

(3) Bond Strength Tests

Tests were conducted to evaluate the bond strength between Grout Slurry No. 5 (the slurry selected in an earlier study - see page 8) and the following:

(a) Polyethylene plastic pipe.

(b) Two small-scale Expendable Grout Retainers and steel casing, coated with grout slurry.

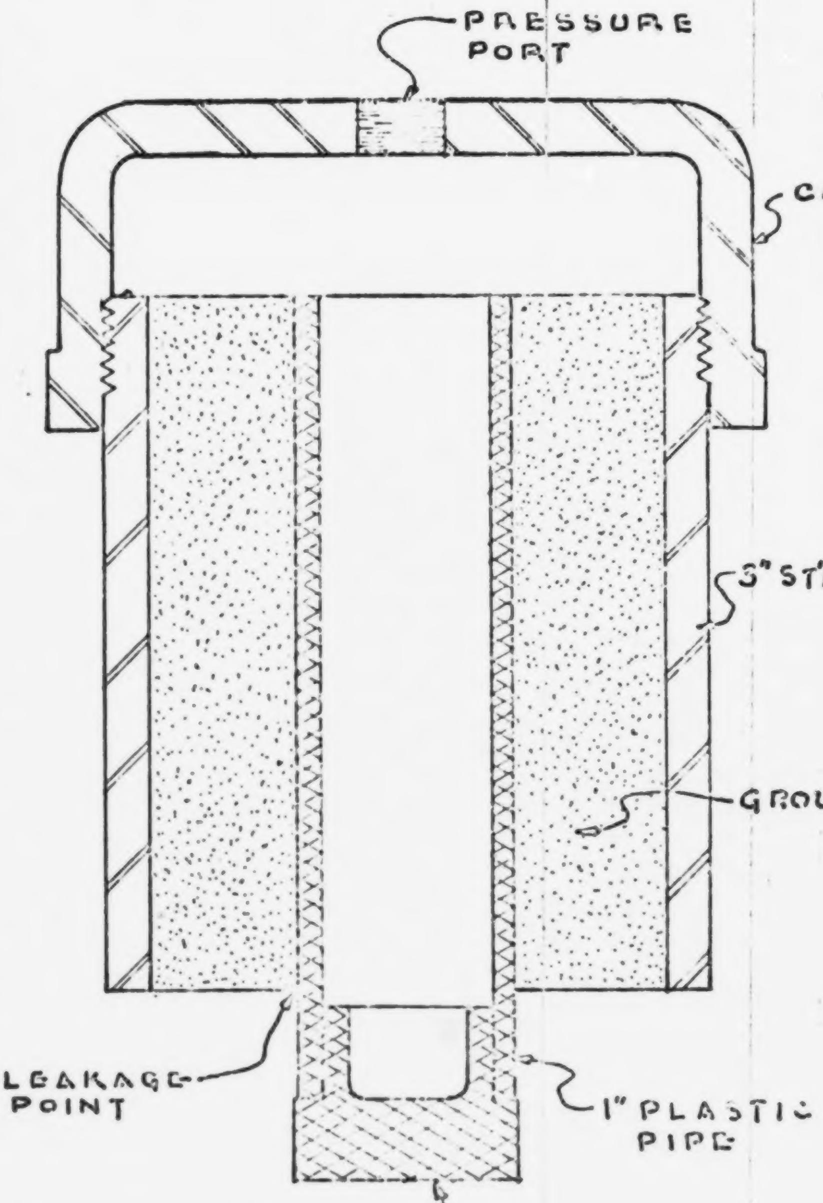
(c) Two small-scale Expendable Grout Retainers and steel casing, uncoated except from leakage through cloth.

For comparison, tests were also run with a grout slurry of Type I Portland Cement containing 5.2 gallons of water to 94 pounds of cement, denoted as Grout Slurry No. 12.

Bond strength was defined as the minimum hydraulic pressure at which leakage occurred between the grout and bonded material when water pressure was applied. Since the purpose of these tests was to arrive at allowable design properties for use in drift plug design, hydrostatic head resistance was used for bond strength rather than pounds of force per square inch of contact, as usually expressed.

Test (a) - Polyethylene Plastic Pipe

Equipment used in these tests is shown in Figure 15. Grout Slurry No. 5, containing 20 parts Type I Cement, 40 parts Fly Ash, 40 parts Gypsum Cement plus 0.6% HR-7 Retarder and 60


PRESSURE PORT
CAP
3" ST'D. PIPE
GROUT
LEAKAGE POINT
1" PLASTIC PIPE
CAP

TEST FIXTURE - BOND STRENGTH

FIGURE 15

parts water by weight, and Grout Slurry No. 12, containing 100 parts Type I Cement and 46 parts water by weight, were tested with one-inch Polyethylene pipe in the test fixture. Tests were conducted at intervals of 24 hours, 48 hours and 72 hours by applying hydrostatic pressure through the pressure port. The minimum pressure at which leakage occurred was then recorded. Allowable design bond strengths of other grout slurries were calculated based on their comparative compressive strengths and are shown in graphic form in Figure 16 along with Slurries No. 5 and 12.

Test (b) - Two Small-scale Expendable Grout Retainers and Steel Casing, Coated with Grout Slurry

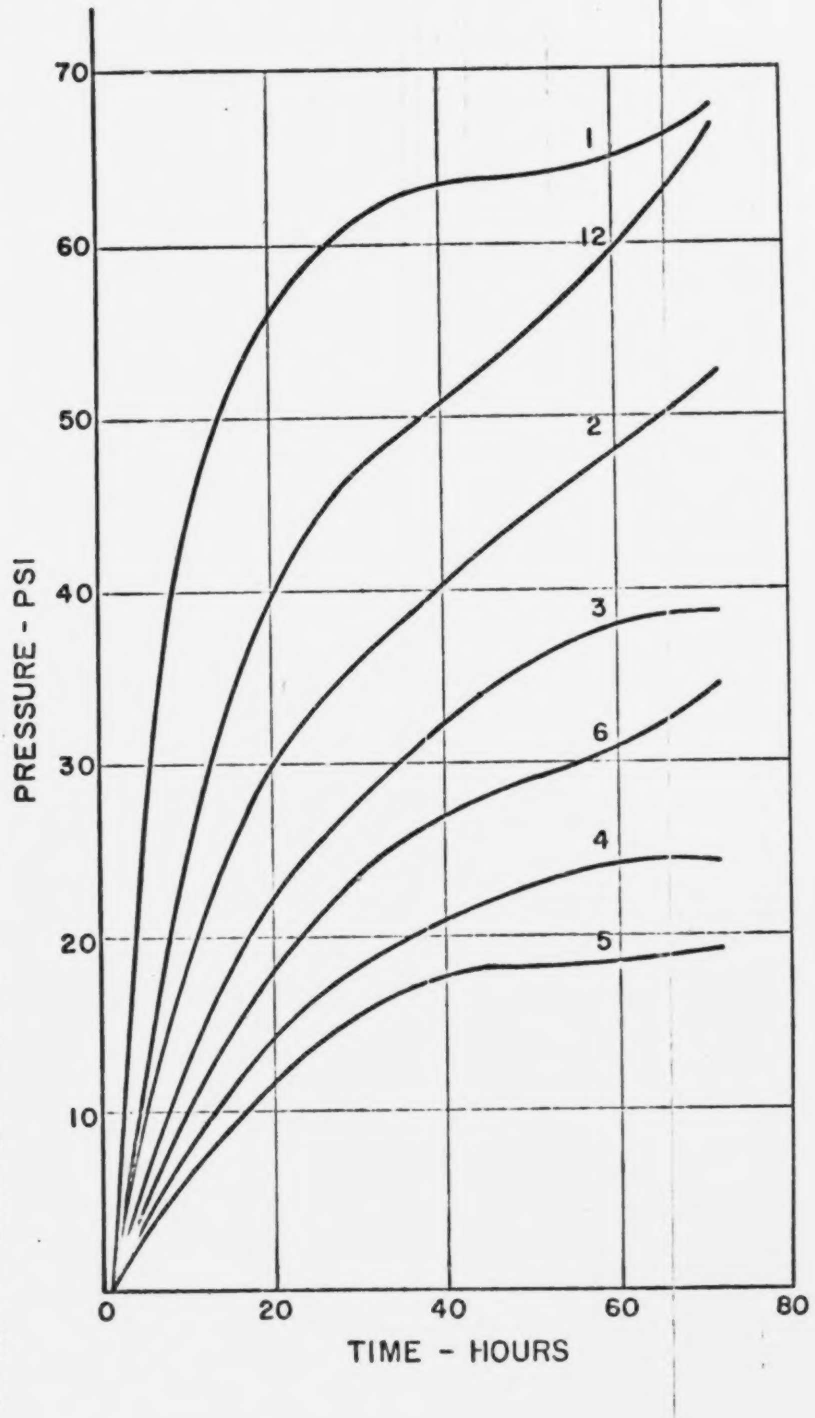
These tests were run using Grout Slurry No. 5 and Grout Slurry No. 12 as used in the previous test. Hydrostatic pressure was applied 72 hours after placement.

Grout Slurry No. 5 showed leakage between retainer and casing at 5 psi but did not leak between retainers until pressure was 30 psi. No attempt was made to measure the amount of leakage, since the primary interest was the point of failure of the bond.

Equipment used in this and the following test is shown in Figure 17.

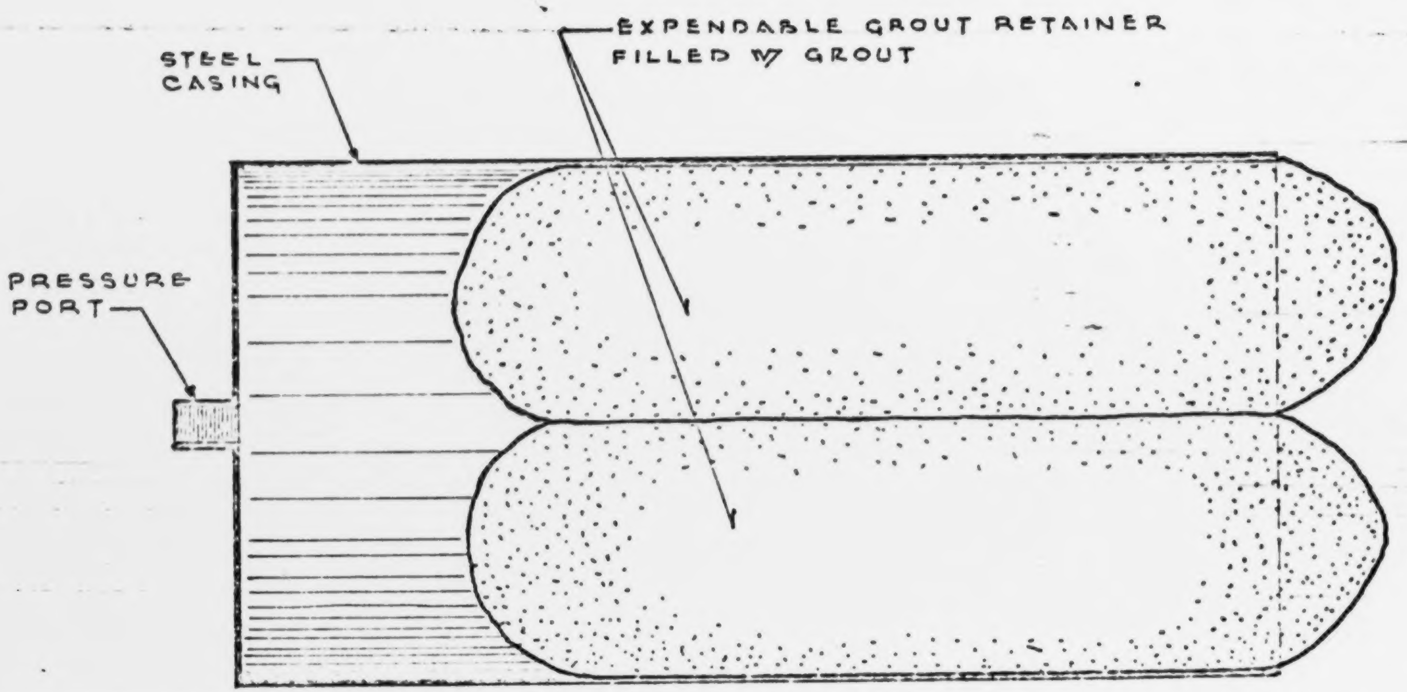
Test (c) - Two Small-scale Expendable Grout Retainers and Steel Casing, Uncoated Except From Leakage Through Cloth

These tests were also run using Grout Slurry No. 5 and Grout Slurry No. 12 as described above. Hydrostatic pressure was



BOND STRENGTH COMPARISON
FOR GROUTS #1 THRU #6 & #12

FIGURE 16


EXPENDABLE GROUT RETAINER
FILLED W/ GROUT
STEEL CASING
PRESSURE PORT

SECTION – BOND STRENGTH TESTS

FIGURE 17

applied at 72 hours after placement. A photograph of this test is shown in Figure 18.

Grout Slurry No. 5 and Grout Slurry No. 12 both showed leakage at below 10 psi between retainers as well as between casing and retainers.

A sample of coal from the Pittsburgh seam was obtained from West Virginia and a test was conducted in the laboratory to determine if Slurries No. 5 and 6 would bond to the coal. Tests were made by pouring the slurry on the coal sample and allowing to harden. A good bond was obtained which was sufficient to prohibit breaking the coal and slurry apart manually.

From these tests, it can be concluded that the bond strength is increased by coating the retainers and surrounding area with the grout slurry as they are placed in the opening. It is also concluded that since the bond strength will increase with age, the strength should be sufficient to withstand a head of water behind the plug by the time the water has impounded sufficiently to create as much as 10 psi.

FIG. 18

(4) Full-scale Grout Retainer Tests:

The objective of this phase of the program was to determine the feasibility of filling a full-scale Grout Retainer with aggregate and grouting in place and to test the improved Expendable Grout Retainer in a full-size simulated mine opening to determine the following information:

(a) The placement techniques involved.

(b) The force exerted on the walls of the mine by the retainers.

(c) The temperature developed by the hydration process of the cement slurry when used in the large quantities contained in the full-size retainer.

Preparation Procedure

A simulated mine drift was constructed in the outside laboratory test area. The drift was constructed of a welded framework of 4-inch, 5.4-pound steel channel on 12-inch centers, anchored in a concrete foundation. The framework was covered on the inside walls and top with 3/4-inch plywood sheets to form a tunnel 16 feet long, 8 feet wide and 7 feet high. The exterior construction can be seen in the side view of the fixed wall shown in Figure 19. The opposite wall was build differently in order to place instrumentation in this wall to measure load and deflection caused by the expanding cement in the retainers. The rear half, 8 feet long, was rigidly fixed in concrete similar to that shown in Figure 19.

FIG. 19

The front half of the wall was of similar construction but was composed of a double wall. The outer part was embedded in concrete, but the inner part was made movable by supporting it with hangers from the roof trusses so that instruments could be placed between the two walls to measure the load imposed on the wall by the retainers. A view of the double wall section is shown in Figure 20. The nine load cells used can be seen in place between the two walls. These load cells were designed to measure a load of from 0 to 35,000 pounds each.

The full-scale test was to consist of building a plug in the simulated mine opening by the placement of three Expendable Grout Retainers to form the seal. Instrumentation was used throughout the test to record the wall loading, wall deflection and heat of hydration.

Three improved Expendable Grout Retainers were built in accordance with the drawing attached as Figure 21, using the fabric selected from tests previously described. All three retainers were 10.5 feet in width, but varied in length. The bottom retainer was 16 feet long before filling. The second retainer was 12 feet in length and the third, or top retainer, had a length of 8 feet. Costs ranged from $36.20 for the 16-foot retainer to $23.40 for the 8-foot retainer.

The slurry used in the test was composed of 20 parts Cement, 40 parts Fly Ash, 40 parts Gypsum Cement and .6% HR-7

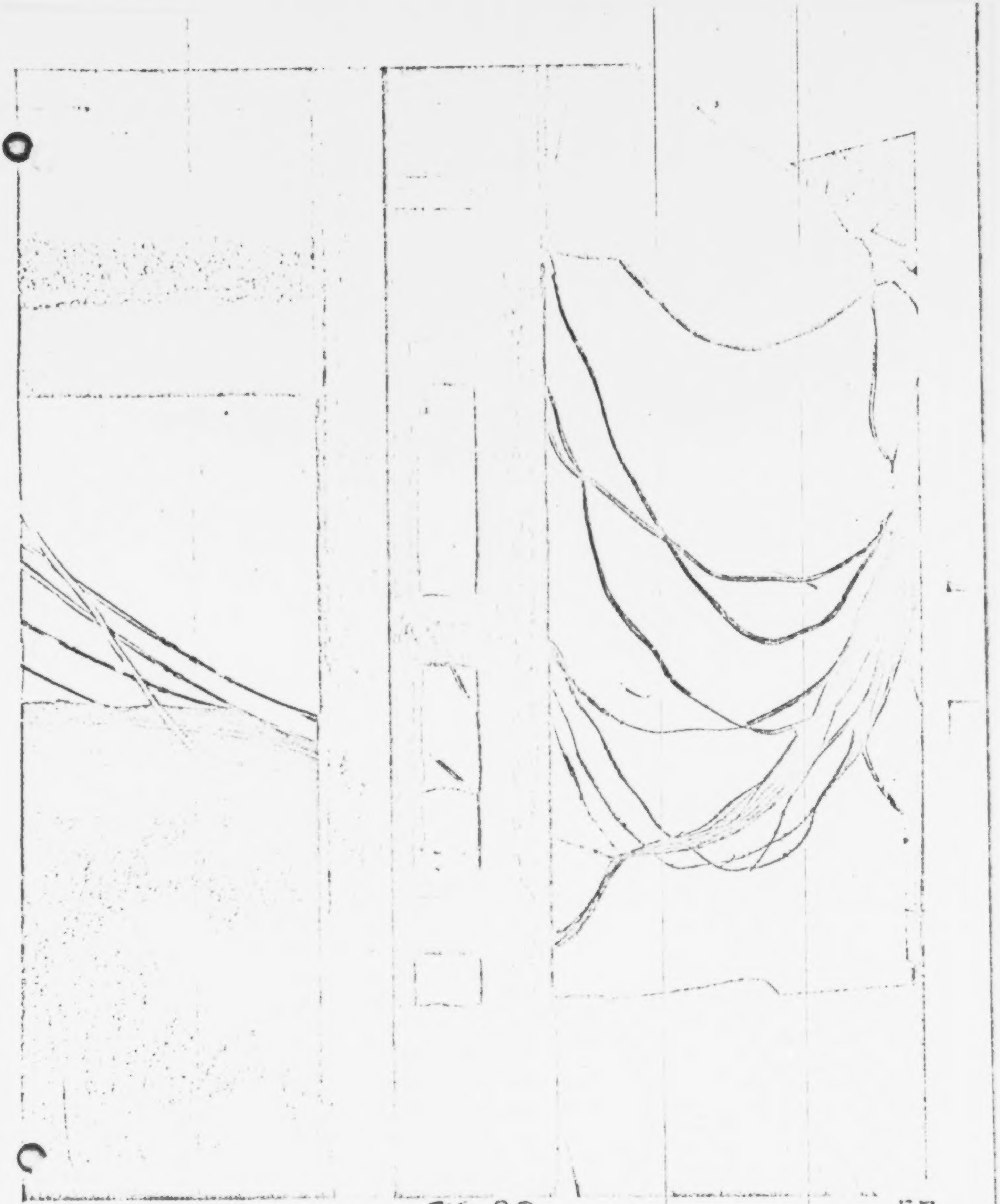
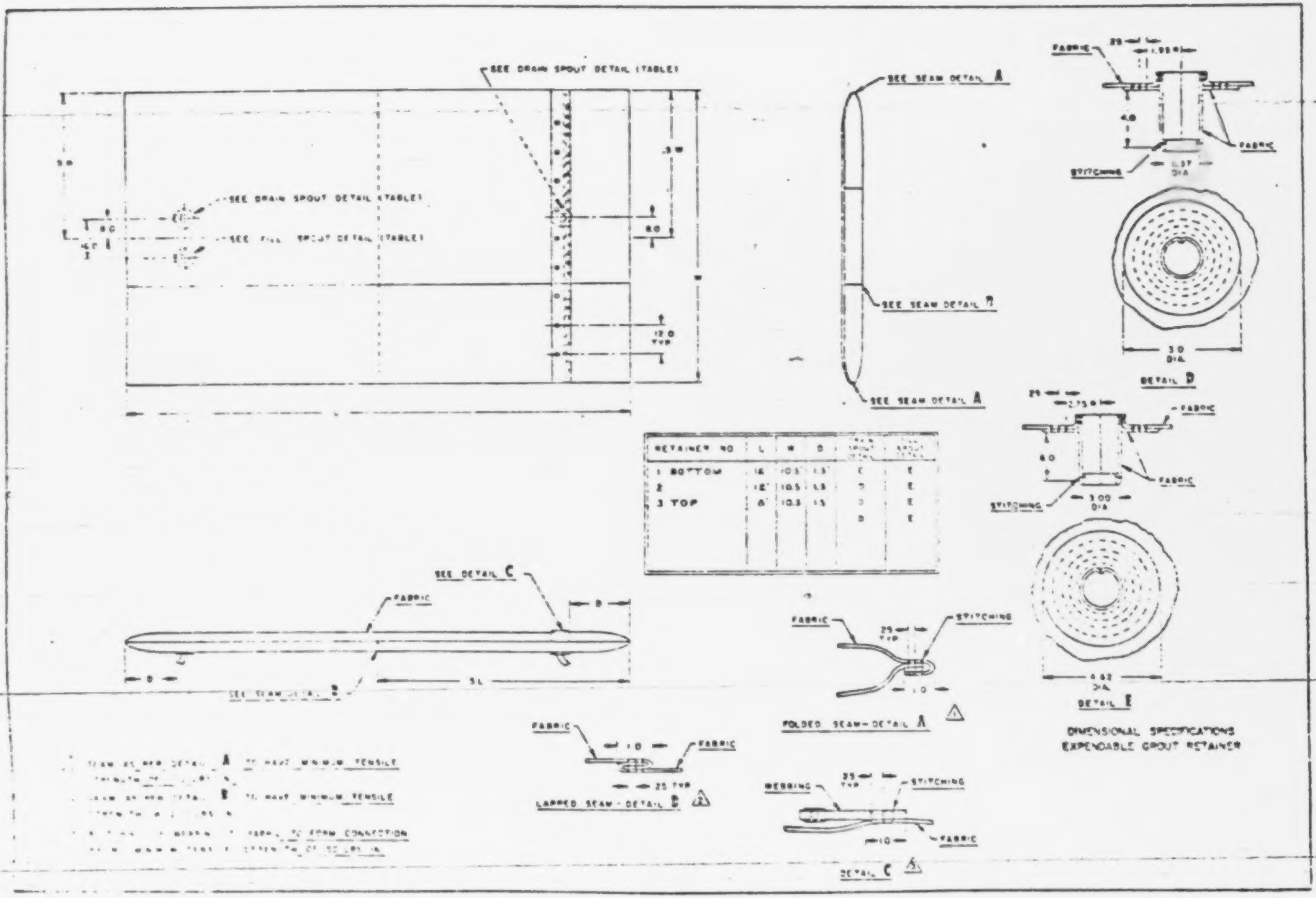


FIG. 20


SEE DRAIN SPOUT DETAIL (TABLE)
SEE DRAIN SPOUT DETAIL (TABLE)
SEE FILL SPOUT DETAIL (TABLE)
.5W
8.0
12.0 TYP
W
L
SEE SEAM DETAIL A
SEE SEAM DETAIL B
SEE SEAM DETAIL A
SEE DETAIL C
FABRIC
SEE SEAM DETAIL B
D
D
.5L
FABRIC
1.0
FABRIC
.25 TYP
LAPPED SEAM - DETAIL B
FABRIC
.25 TYP
STITCHING
1.0
FOLDED SEAM - DETAIL A
WEBBING
.25 TYP
STITCHING
1.0
FABRIC
DETAIL C
FABRIC
4.0
FABRIC
STITCHING
1.37 DIA
3.0 DIA
DETAIL D
2.75
FABRIC
6.0
FABRIC
STITCHING
3.00 DIA
4.62 DIA
DETAIL E

RETAINER NO	L	W	D	DRAIN SPOUT DETAIL	FILL SPOUT DETAIL
1 BOTTOM	16'	10.5'	1.5'	D	E
2	12'	10.5	1.5	D	E
3 TOP	8'	10.5	1.5	D	E
				D	E

1. SEAM AS PER DETAIL A TO HAVE MINIMUM TENSILE STRENGTH OF [illegible]
2. SEAM AS PER DETAIL B TO HAVE MINIMUM TENSILE STRENGTH OF [illegible]
3. [illegible] WEBBING [illegible] FABRIC TO FORM CONNECTION [illegible] STRENGTH OF [illegible] LBS/IN

DIMENSIONAL SPECIFICATIONS
EXPENDABLE GROUT RETAINER

Fig 12.1

Retarder and 60 parts water by weight. This slurry was selected in the test work described in Section A of this report.

Placement Technique

The bottom retainer was placed in position in the simulated tunnel after covering the floor and lower side walls with grout slurry. Ropes were laced through the eyelets on the webbing strap affixed to the retainer. These ropes were then passed through six pulleys attached to an anchoring device developed for this test. The anchoring device is an L-shaped steel platform which can be weighted with sand bags to restrain any longitudinal movement of the retainer while being filled. A thermocouple was placed on the ground under the retainer to measure the temperature of hydration.

Figure 22 shows the front of the bottom retainer after filling, with the anchoring device in place.

The retainers were filled with the designated slurry through a 2-inch line using a centrifugal pump capable of pumping 600 gallons per minute at 30 psi pressure. The first retainer was expanded to a height of 30 inches in 30 minutes. After a waitimg time of one hour, the retainers had hardened sufficiently to allow preparations to begin for placement of the second retainer. The anchoring ropes were removed from the retainer in place by unlacing the rope from the eyelet strap. A strain gage, designed especially for use when embedded in concrete was installed between

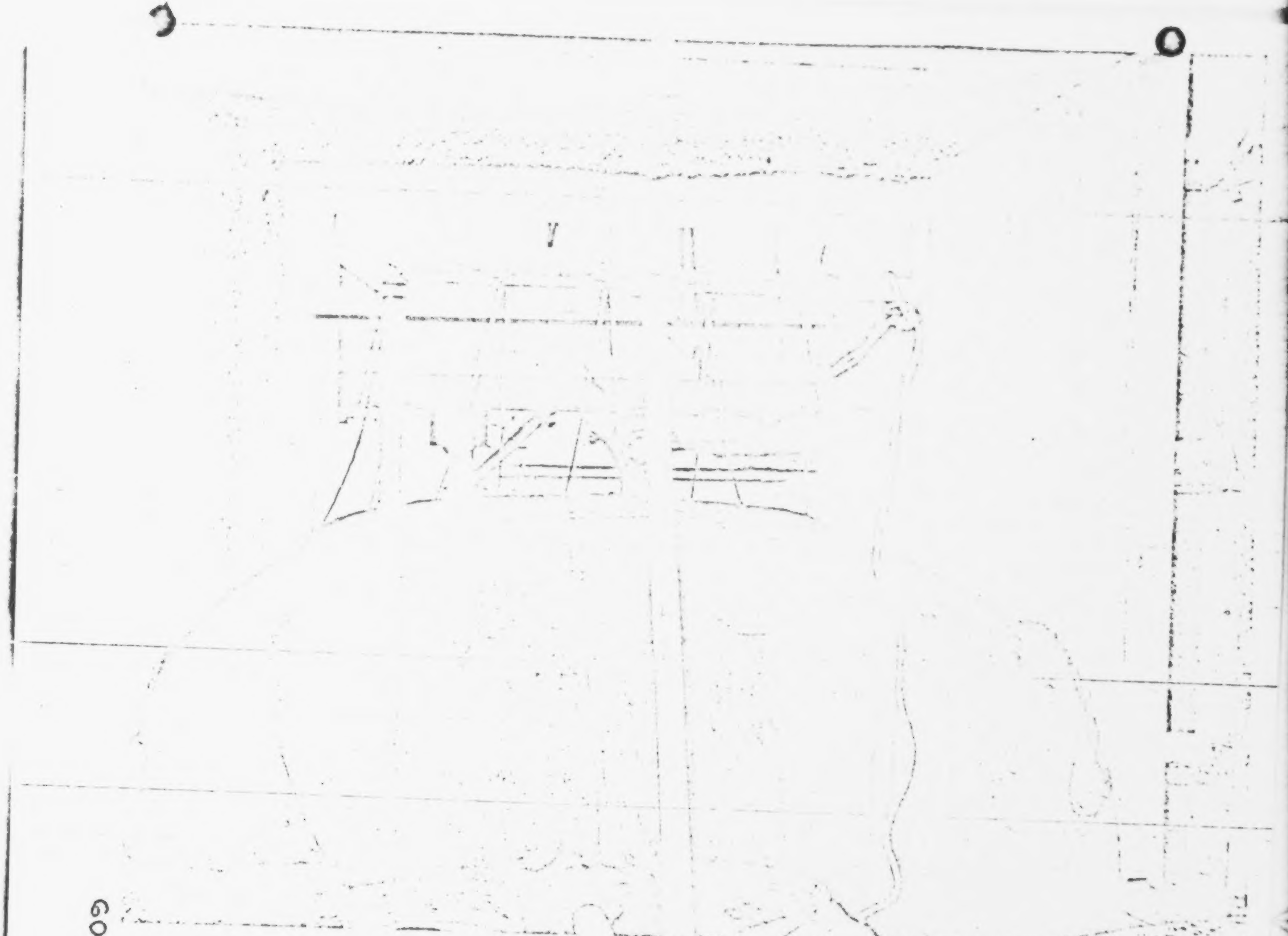


FIG. 22

the first and second retainers and embedded in the slurry sprayed over the top of the bottom retainer. Another thermocouple was also placed between the retainers. The second retainer was put in place and the anchor ropes attached to the webbing strap. The pump was connected to the retainer and pumping commenced two hours after the bottom retainer was finished. Time required to fill the second retainer to its height of 30 inches was 30 minutes. Waiting time for this retainer was 2 hours, part of which was used in preparing for placement of the third, or top, retainer. Preparation was similar to that of the other retainers, including placement of a thermocouple above and below this retainer, except that the anchor system was not utilized for the top retainer.

Only 20 minutes' pumping time was required to bring the retainer into contact with the roof. It was necessary to continue to pump at short intervals during the next 30 to 40 minutes to replace the void spaces created by the water loss through the cloth in order to keep the retainer in contact with the roof. This required keeping a maximum pressure of 5 psi on the retainer during this process. A photo of the finished plug is shown in Figure 23.

The technique used in the placement of this mine opening plug is a definite improvement over that previously used from a cost reduction standpoint, since it is possible to place this entire plug in one 8-hour day, using a minimum amount of equipment.

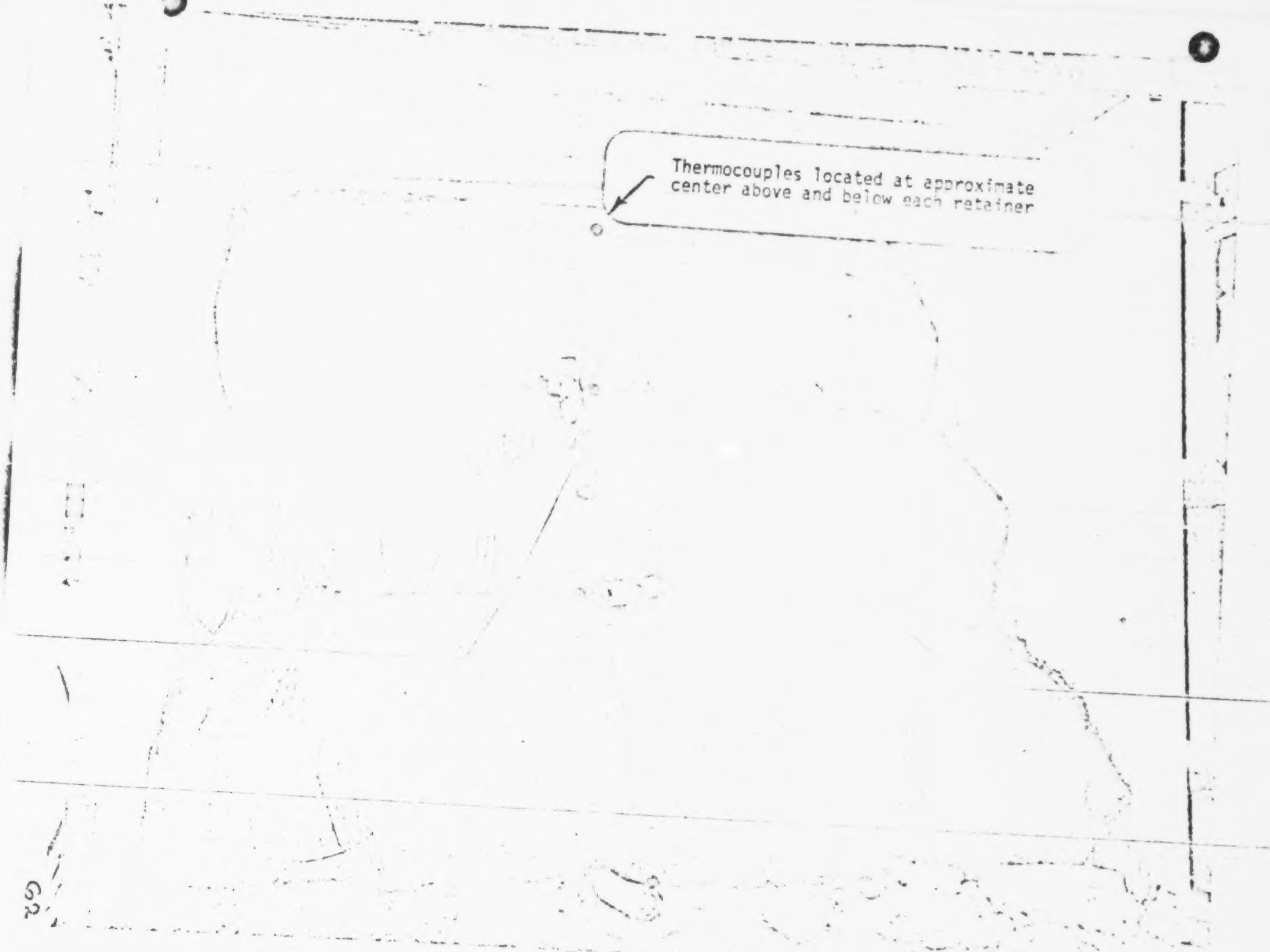

Thermocouples located at approximate center above and below each retainer

FIG.23

Instrumentation

Various types of instrumentation were incorporated into the test to obtain the desired information. The strain gage, embedded between the first and second retainers to determine the strain which took place as the grout slurry expanded, did not function. Apparently the protective covering on the strain gage leaked and caused the gage to become inoperative, so no data was obtained from this installation.

Instrumentation installed for measurement of the wall deflection consisted of a displacement transducer capable of measuring a movement of ± 0.10 inches with an accuracy of ± .5% of full-scale on the readout equipment.

The wall displacement was measured by the transducer mounted on a spring-loaded steel bar between two fixed portions of the walls. Any wall movement resulted in an increased voltage output for a constant power voltage which was indicated on a voltmeter. A close-up view of the transducer installation is shown in Figure 24. The results of the test indicate that the major portion of the deflection came during the first six hours of the retainer installation when the deflection went from 0 to .210 inches. As can be noted in Figure 25, a displacement of only .073 inches occurred from the sixth hour until 170 total elapsed hours, but this was a constant increase for the time monitored.

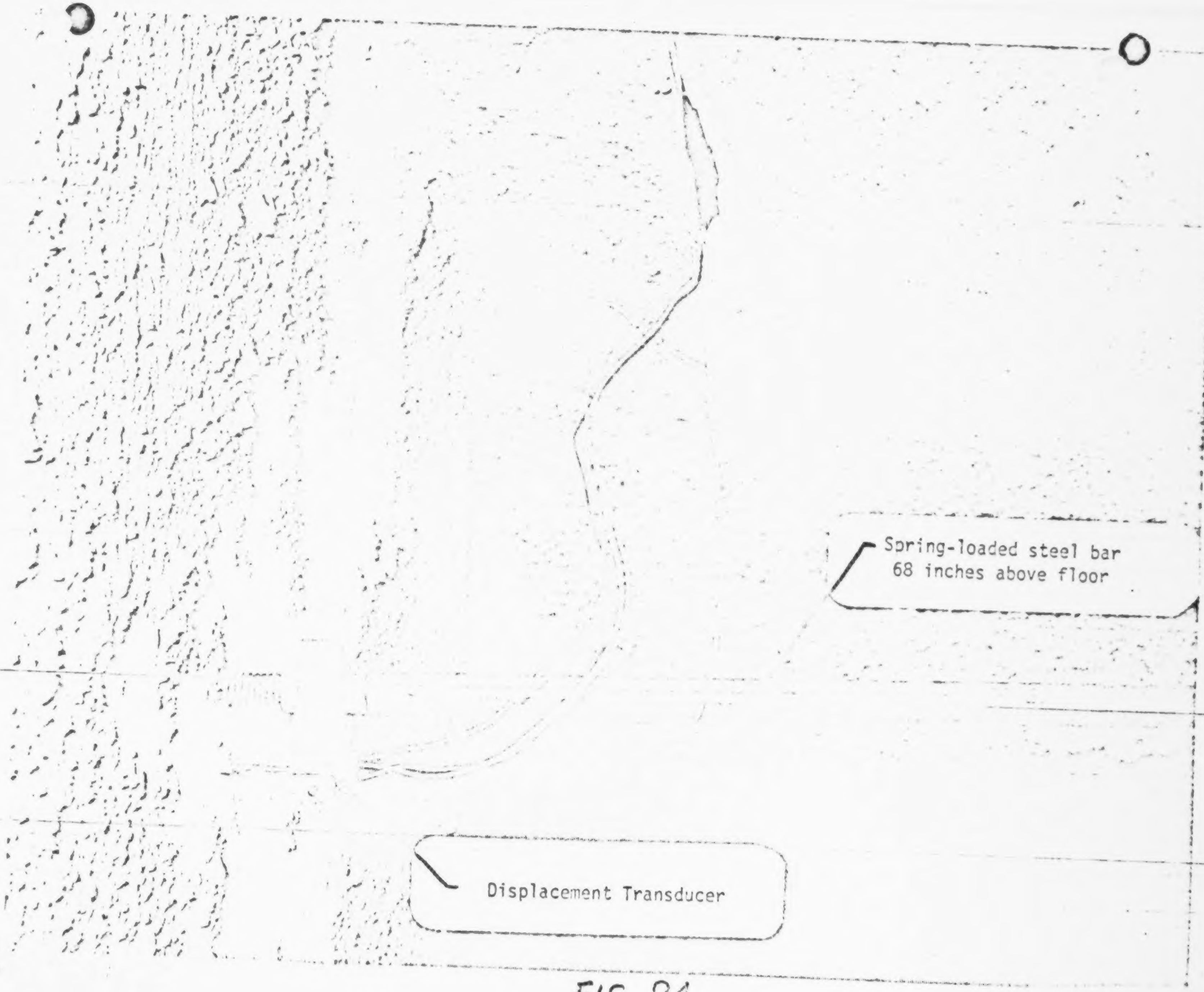
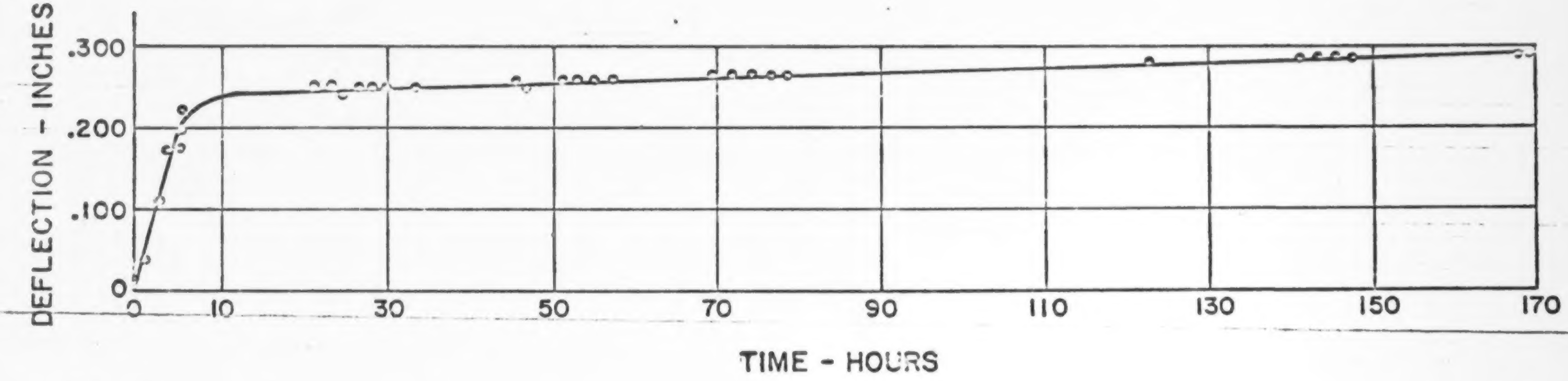

Spring-loaded steel bar
68 inches above floor
Displacement Transducer

FIG. 24

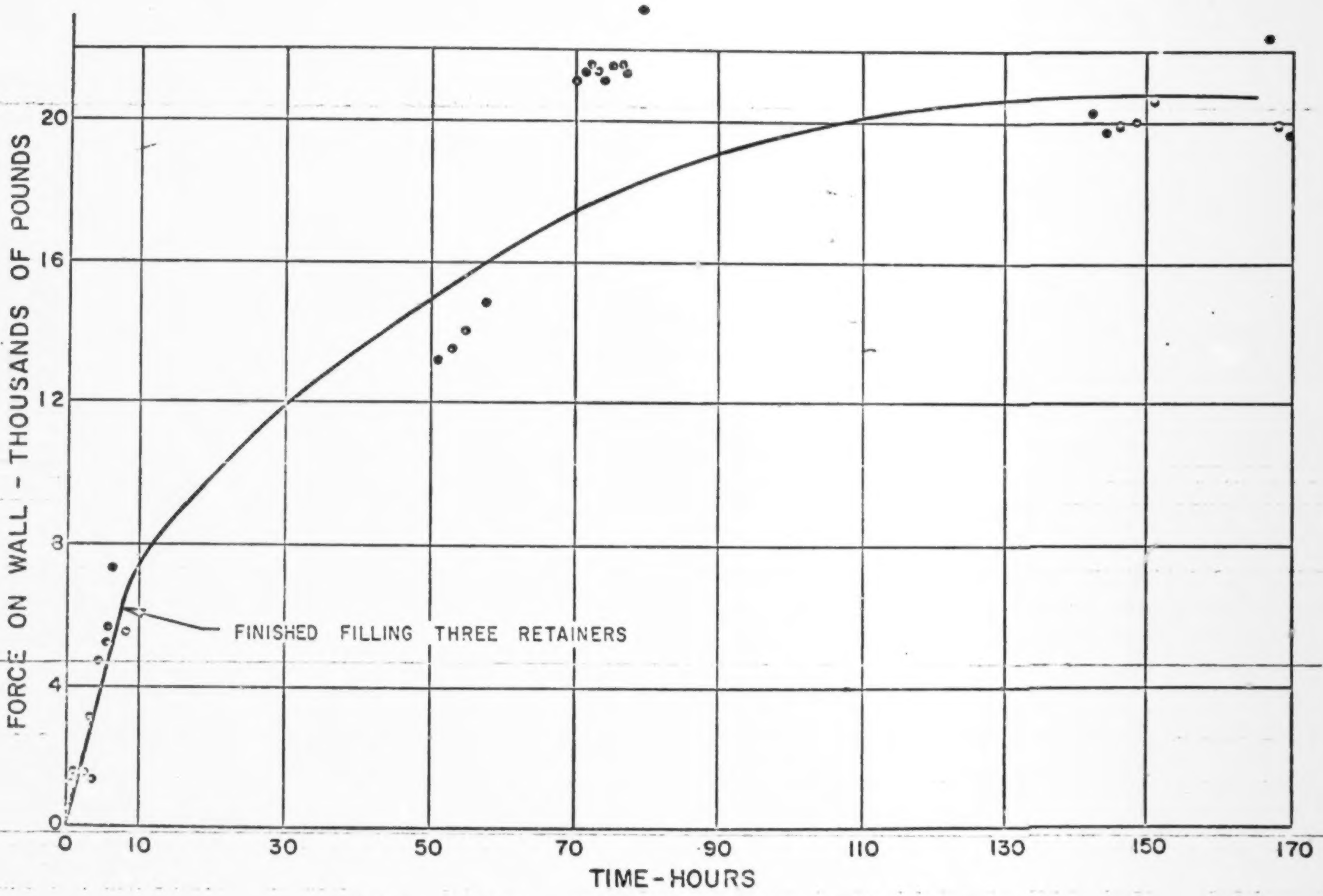


TUNNEL WALL DEFLECTION
FULL SCALE TEST

FIGURE 25

The force on the movable wall was recorded by nine load cells placed between it and a fixed portion of the wall. The force measured by these load cells is shown on the curve in Figure 26. The maximum force of 21,750 pounds was exerted on the wall 170 hours after the start of the test. On the basis of 34.5 square feet of contact area for the retainers on the wall, this resolves to a force of 630 pounds per square foot.

Several observations could be made based on an interpretation of the above test results. The rigidly fixed side walls resemble a beam fixed securely at each end. The bending of the wall, as indicated by the tranducer data, took place primarily during the installation of the retainers. The weight of the grout slurry in the retainers caused a deflection amounting to 74% of the total deflection. The other 26% of the deflection came gradually over a period of approximately 160 hours due to expansion of the grout slurry in the retainers. The load on the movable wall, recorded from the load cells, went to about 6,000 pounds during the 6 hours required for placement of the retainers. The data indicated a maximum load of 21,750 pounds at 75 hours after start of test, and again at 170 hours. The data is somewhat erratic and could be in error; however, due to the nature of the test, it was impossible to rerun the test. Therefore, the data must be interpreted in view of the results of the deflection test and the laboratory expansion tests.



TUNNEL WALL LOADING - FULL SCALE TEST

FIGURE 26

Both of these tests indicate that the expansion of the cement grout occurs as a gradual increase for at least the 170 hours monitored during this test. Based on this data, points were plotted and the curve for the load tests was interpolated and is shown in Figure 26. The readout instruments used in these tests are shown in Figure 27, along with the recording pyrometer used to receive information from the Iron-Constantan thermocouples installed on the retainers.

The results of the measurement of hydration heat from the cement grout are shown in Figure 28. A maximum temperature of 160° F. was recorded after approximately 140 hours. This was obtained from a thermocouple mounted between two retainers. This compared to a maximum temperature of 131° F. in 16.7 hours obtained with the same grout slurry in laboratory tests. The thermocouples placed on bottom and top of plug showed much lower temperatures. This was probably caused by exposure to ground and to air by bottom and top retainer, respectively. The hydration heat values obtained are not high enough to cause any detrimental effects in the plug or surroundings.

Strength Tests

Samples of the grout slurry were taken from the mixer sump during the filling of the retainers and allowed to cure for compressive strength tests. Cores were also cut from the hardened retainers for compressive strength tests. Tests were made after

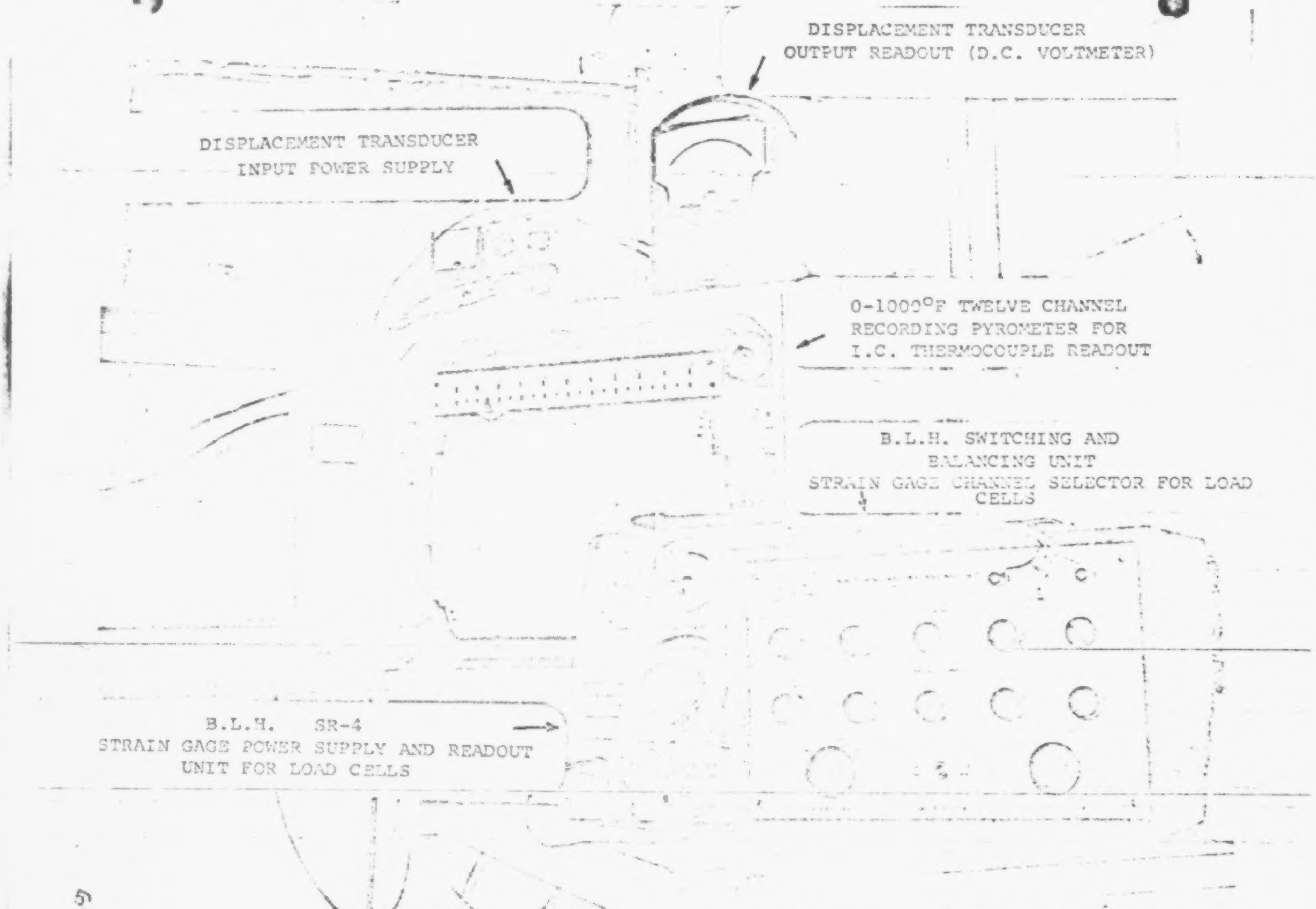
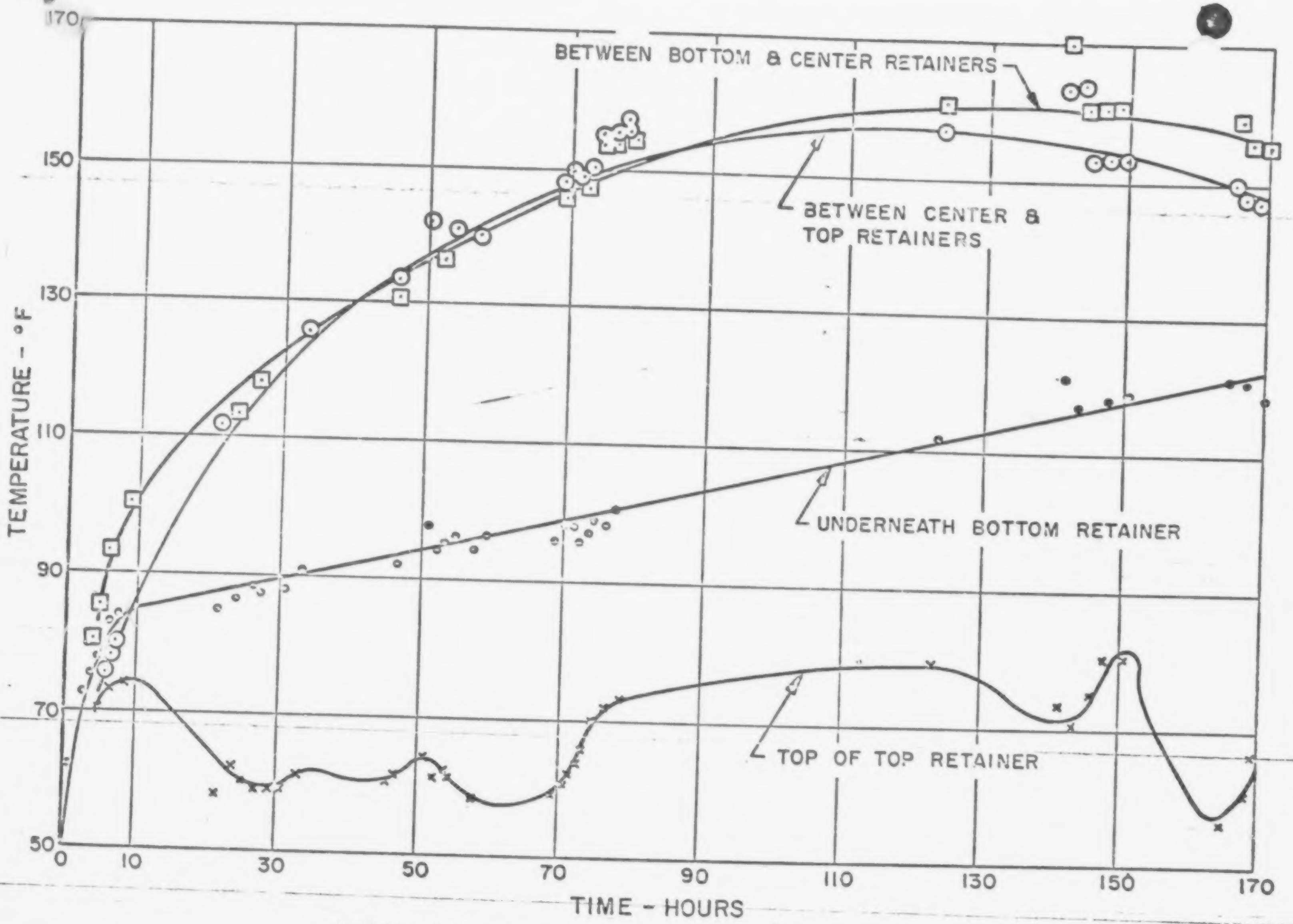

DISPLACEMENT TRANSDUCER OUTPUT READOUT (D.C. VOLTMETER)
DISPLACEMENT TRANSDUCER INPUT POWER SUPPLY
0-1000°F TWELVE CHANNEL RECORDING PYROMETER FOR I.C. THERMOCOUPLE READOUT
B.L.H. SWITCHING AND BALANCING UNIT STRAIN GAGE CHANNEL SELECTOR FOR LOAD CELLS
B.L.H. SR-4 STRAIN GAGE POWER SUPPLY AND READOUT UNIT FOR LOAD CELLS

FIG. 27



HEAT OF HYDRATION - FULL SCALE TEST

FIGURE 28

curing times of 24 hours, 48 hours and 9 days. These test results are shown in Table VIII. Although the samples from mixer sump for the bottom retainer (Part A of table) only showed 14 psi strength, the cored samples of hardened material in the retainer had compressive strengths of 117 and 185 psi. The figures for compressive strength also varied within the retainer, with samples cored from the bottom section showing strength as high as 280 psi. This indicates that a settling of the solids occurred to give this portion greater strength. As a result of this finding, a change was made in the recommended slurry from a water content of 60 parts, used in these tests, to 41 parts. This is a change from Grout Slurry No. 5 to Grout Slurry No. 6. The new slurry recommended gives a density increase affording less settlement of solids and therefore a more consistent and higher strength grout throughout the retainer.

Laboratory tests indicated that aggregate would fill about 55% of a container, leaving 45% to be filled with grout. This would provide economic advantages in filling an Expendable Grout Retainer. Therefore, a full-scale test was performed using this method.

An Expendable Grout Retainer measuring 10 feet in length and 7 feet in width was filled pneumatically with 3/8-inch graded limestone chips. Techniques developed in tests discussed in Section E were used in placement of the aggregate. Equipment used

TABLE VIII

GROUT COMPRESSIVE STRENGTH

FULL SCALE TESTS

Sample Location	Curing Time	
	24 Hrs.*	48 Hrs.**
A. Sample From Mixer Sump When Filling Bottom Retainer: (Set Density - 13.66 lbs./gal.)	14 psi	14 psi
B. Samples Cored From Hardened Retainers:		
Bottom Retainer - Bottom Section	185 psi	117 psi
Middle Retainer - Bottom Section	125 psi	229 psi
Top Section	53 psi	---
Top Retainer - Bottom Section	280 psi	170 psi

C. Samples Cored From Hardened Retainer 9 Days After Mixing: *

Sample Location	Retainer		
	Bottom	Middle	Top
Mixer Sump	127 psi	140 psi	375 psi
Top Part of Retainer	260 psi	1950 psi	720 psi
Center of Retainer	---	1596 psi	770 psi
Bottom of Retainer	725 psi	---	1308 psi

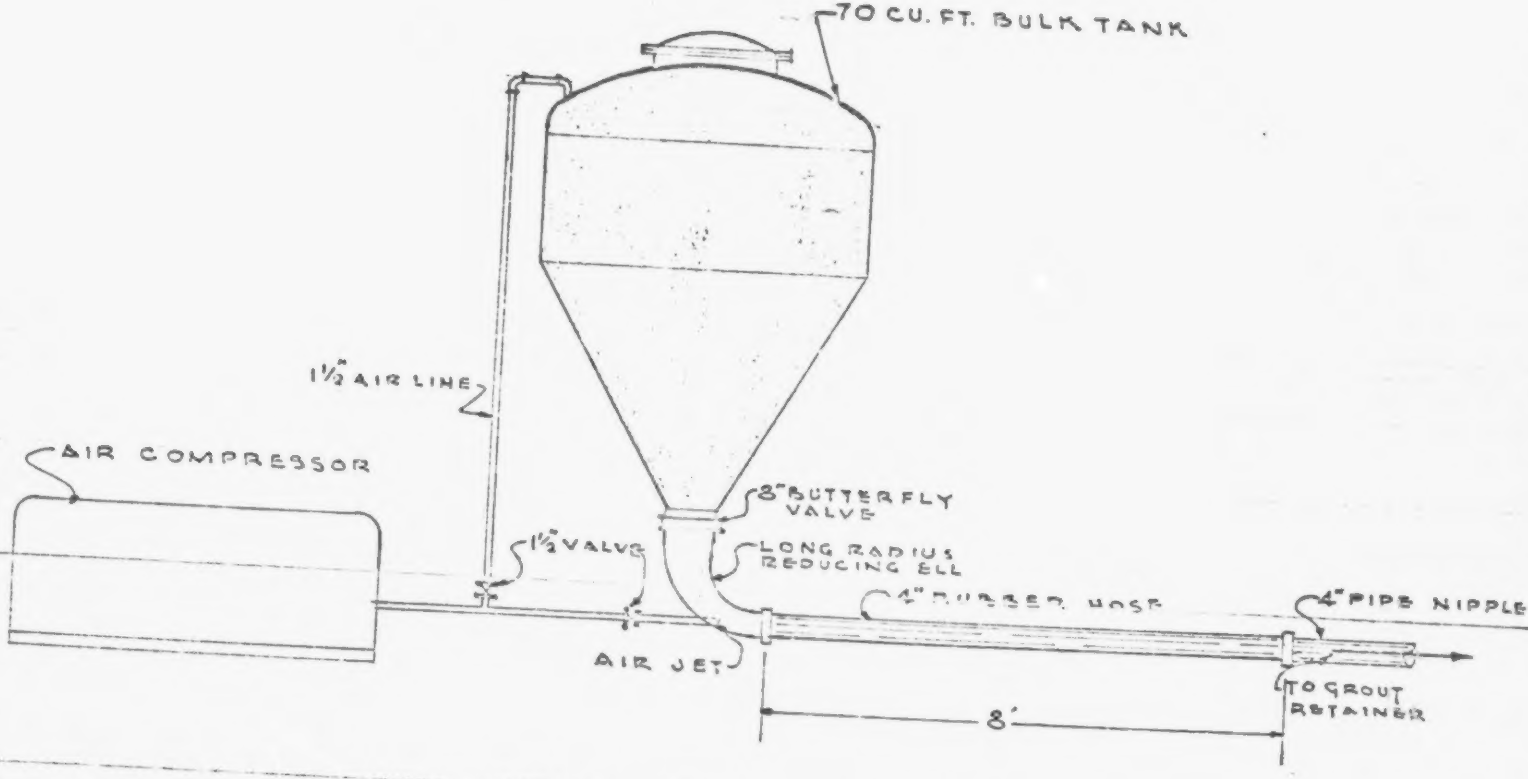
* Cured Outside at 40° F. to 60° F.
** Cured Inside for Last 24 Hours at 70° F.

is shown in Figure 29. A pressure of 20 psi was placed on the aggregate in the tank to move the material. The main problem encountered was that of directing the aggregate placement to obtain an even fill in the retainer. This can be noted in Figure 30 by the shape of the retainer after filling with aggregate.

At this point, a 2-inch grouting pipe containing 19 1/2-inch holes on 12-inch centers was driven into the aggregate. The grout pipe was oriented so that the grout holes were horizontal in the retainer.

In prior laboratory tests it was determined that the aggregate would contain voids sufficient to hold 3.75 gallons per hundred pounds of aggregate. Since 10,000 pounds of aggregate were blown into the retainer, the voids should have contained 375 gallons of grout. In addition, it was assumed that the outer part of the retainer would hold additional grout, so provision was made to mix a larger quantity than required for the voids in the aggregate. The grout slurry used was Grout Slurry No. 2 composed of 56 parts Cement, 44 parts Fly Ash, 4 parts Calcium Chloride and 50 parts water by weight. A total of 600 gallons of grout was pumped into the retainer at a maximum rate of 80 gallons per minute.

After the slurry had hardened for six days, the retainer was cut and stripped away from the mass of grout and aggregate.


70 CU. FT. BULK TANK
1½" AIR LINE
AIR COMPRESSOR
8" BUTTERFLY VALVE
1½" VALVE
LONG RADIUS REDUCING ELL
4" RUBBER HOSE
4" PIPE NIPPLE
AIR JET
TO GROUT RETAINER
8'

TEST SET-UP
AGGREGATE PLACEMENT IN RETAINER

FIGURE 29

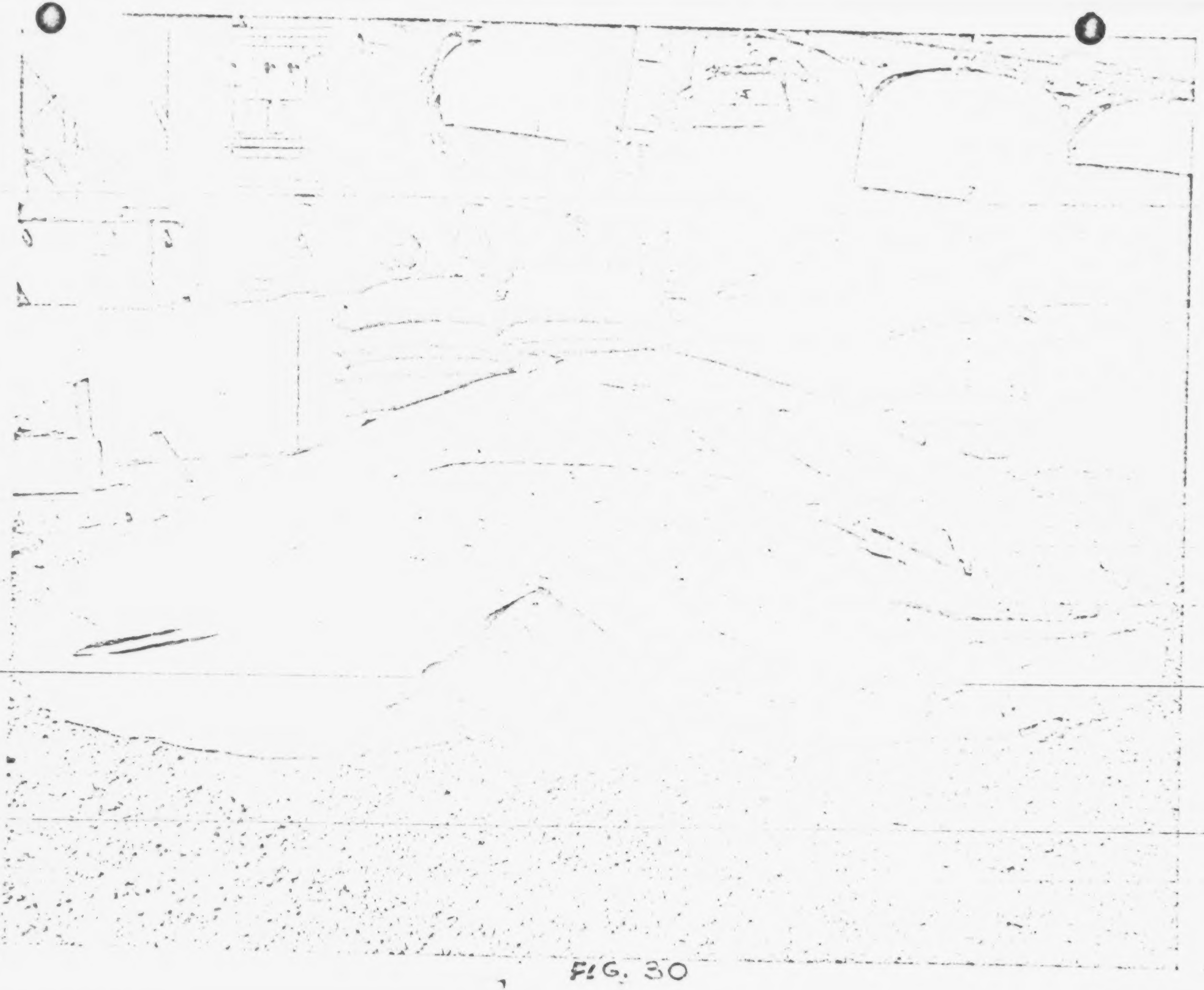


FIG. 30

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best available copy.

Figure 31 shows the retainer at this time. A large amount of unconsolidated aggregate can be seen in the center of the retainer. The consolidated portion left after removal of the unconsolidated material is shown in Figure 32. The area adjacent to each grout hole and the area near the entrance of the pipe into the retainer seems to be well consolidated with grout. The grout also formed a shell conforming to the shape of the retainer, but this shell would not have been capable of supporting the weight of another retainer. An examination of the aggregate revealed the presence of very fine material mixed with the larger sizes of rock. This would have prevented the proper distribution of grout slurry through the aggregate.

These full-scale tests showed that the retainers apply a sustained load over the floor and walls of a drift. The technique of slowly pumping into the top retainer or maintaining a slight pressure on it is necessary to accomplish the desired seal at the roof of the drift. It was also determined that a dense grout slurry is necessary to prevent the settlement of solids while the slurry takes its initial set.

Due to difficulty of placing aggregate within a retainer and subsequently grouting, it was concluded that this approach would not provide an economical advantage over using all grout slurry since extra equipment would be involved for placement. However, it might be feasible to mix the slurry and aggregate as a concrete mix and fill the retainer with this, since this could be accomplished more economically.

FIG. 31

FIG. 32

C. Auger Hole Plug Development

The objective of this phase of the testing was to develop an expendable closure plug suitable for use in auger holes.

Two types of expendable closure plugs were developed and tested. One type was a wooden wiper plug, shown in Figure 33. This plug consisted of two circular wipers separated by wooden stringers. The circular wipers were composed of heavy rubber belting 24 inches in diameter, fastened between two plywood discs of 20-inch diameter. The fill hose through the front disc and the drain hose through the entire plug can be seen in the photograph. The plugs used in this test were 3 feet long and made for testing in 24-inch concrete pipe.

The other type plug tested was a special cloth type retainer designed for use in the round auger holes. The cloth was made into a cylindrical sleeve. It was then banded to a special wooden disc on one end using steel banding material. This disc contained two holes for the 2-inch plastic fill pipe and drain pipe. These pipes were held in place with hose clamps on the pipe adjacent to each side of the disc. The rear side of the plug was closed by gathering the cloth around the drain pipe and banding securely in place.

The first test was conducted using the wooden wiper plug. The initial tests of this plug in a 24-inch diameter concrete pipe indicated that the rubber wipers consisting of

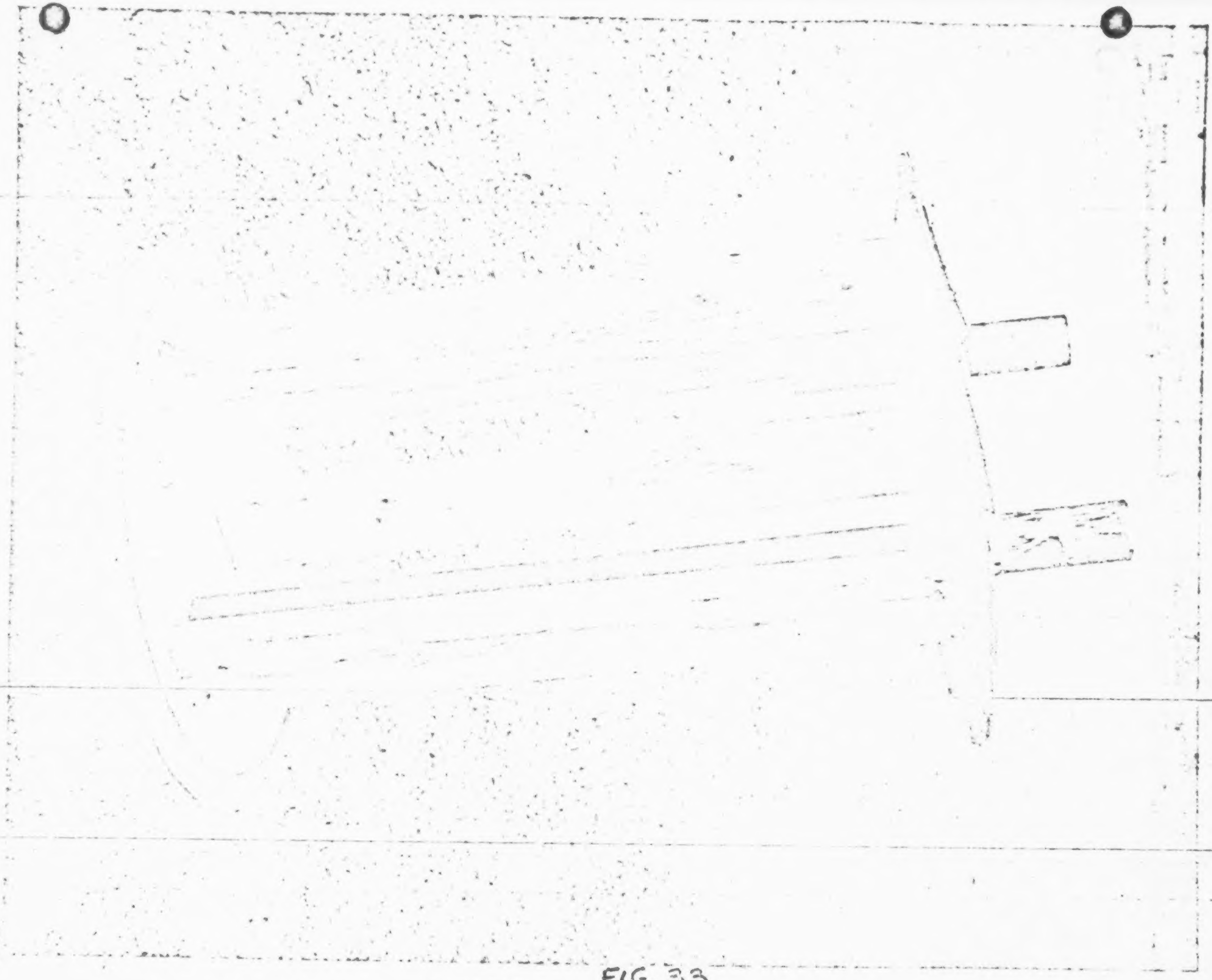

FIG. 33

two layers of 1/8-inch belting were too flexible to hold the head of grout slurry. Heavier wipers were made from 3/8-inch rubber belting and installed on the plug before it was again inserted into 24-inch concrete pipe. In this test, the grout slurry was successfully placed between the wipers. When the cement grout hardened, the expansion of the slurry caused the concrete pipe to crack, as shown in Figure 34. These expansive qualities would aid in providing a good seal in a field application.

The cloth type retainers were also tested in the 24-inch concrete pipe. It was found that it was necessary to keep adding additional grout slurry to compensate for the loss of water through the cloth, but it was possible to completely fill the retainer by maintaining a pressure of 6 psi on the pump until the slurry took its initial set. A view of this test is shown in Figure 35.

The wooden type plug cost approximately $120.00 to construct. The fabric type cost approximately $23.00. Cost of filling each should be comparable. The fabric type would be easier to handle and place in the auger hole.

As a result of these tests, it was decided to construct two wooden type plugs for field tests with the balance of the auger holes to be plugged using the fabric retainer.

FIG. 34

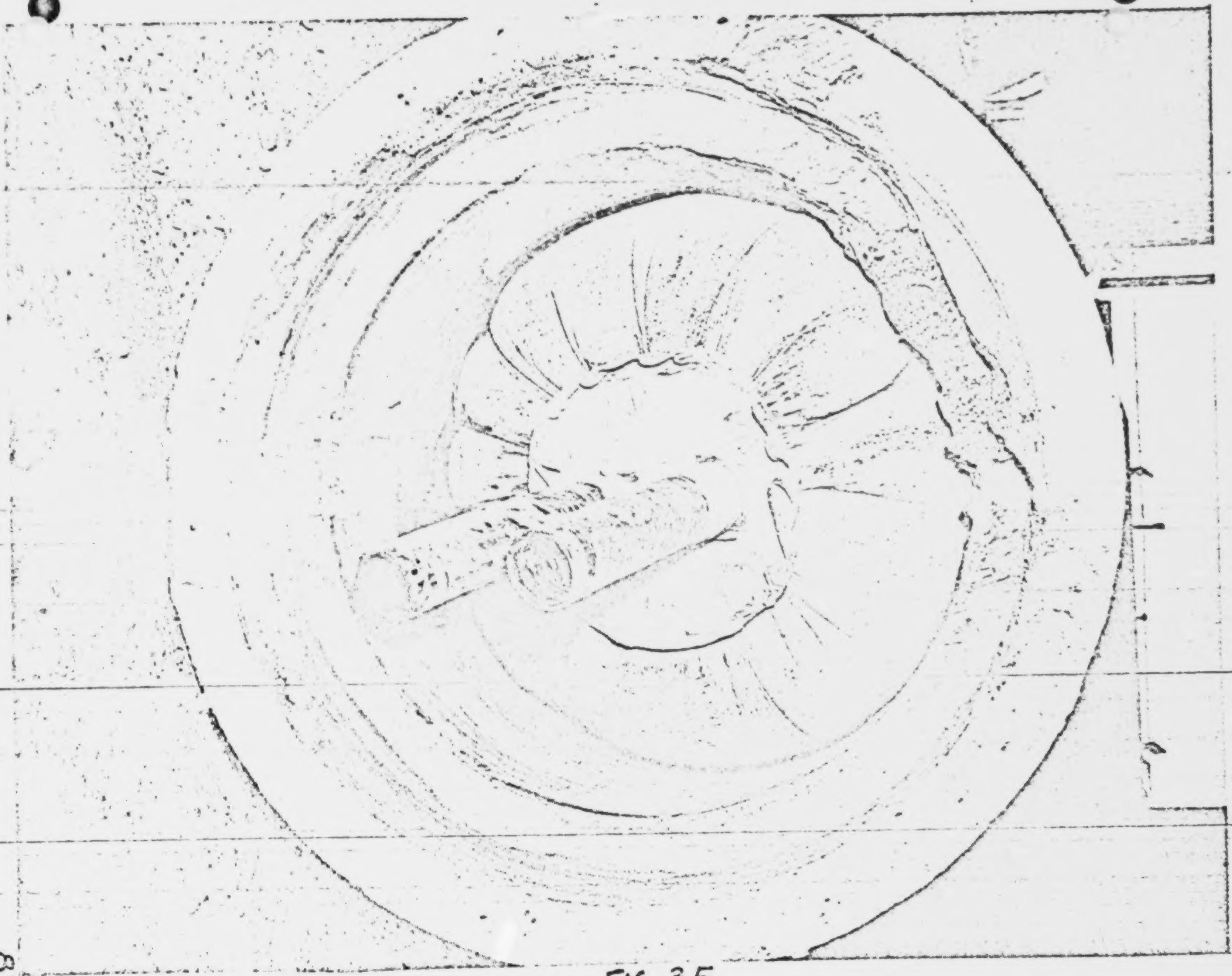

FIG. 35

D. Remote Bulkhead Building

This phase of the experimental work was to determine if a cementitious material could be sprayed on the floor, roof and walls of a mine drift so that the material would form a bulkhead across the mine drift.

Two different types of fast-setting grout slurries were considered and tested. One slurry was a latex cement-gypsum combination. The other slurry was a silicate cement.

Laboratory tests on a small scale were conducted using a different type of quick-setting cement. Latex type gypsum cement slurries were tested for compressive strength and setting time. Results are shown in Table IX. The slurry made with 40 parts Cement, 60 parts Gypsum Cement, 1 part Latex and 45 parts water by weight was used in a spraying test. The slurry was sprayed inside a small cardboard box until a build-up was obtained all around the sides. Spraying was accomplished by using 10 psi air pressure on a chamber containing the slurry. The build-up was obtained by spraying on successive layers each two minutes. A thickness of from 1/2 to 1-1/2 inches was obtained. After five days at room temperature, there were no visible signs of cracking or pulling away from the walls. A photograph of this test is shown as Figure 36.

Samples of the slurry were made in standard 2-inch cube laboratory molds by filling the mold half full from the first mix,

TABLE IX

FAST-SETTING LATEX SLURRIES

70° F.

Ingredients - Parts by Weight					Compressive Strength-Psi		
Cement	Gypsum Cement	Latex*	Water	Set Time (Mins.)	30 Min.	45 Min.	1 Hr.
40**	60	1	55	30	---	575	540
20***	80	1	60	30	---	825	850
40	60	1	62.5	15	582	560	---
20	80	1	72.5	20	520	---	560

Slurries Containing 5% Salt by Weight of Water

Ingredients - Parts by Weight					Compressive Strength-Psi	
Cement	Gypsum Cement	Latex*	Water	Set Time (Mins.)	10 Min.	20 Min.
40	60	1	62.5	3	350	330
40	60	1	55	3	465	475
40	60	1	50	-	675	---
40	60	1	45	2	830	---

* Powdered
** Contained 0.25 Parts HR-7 Retarder
*** Contained 0.1 Parts HR-7 Retarder

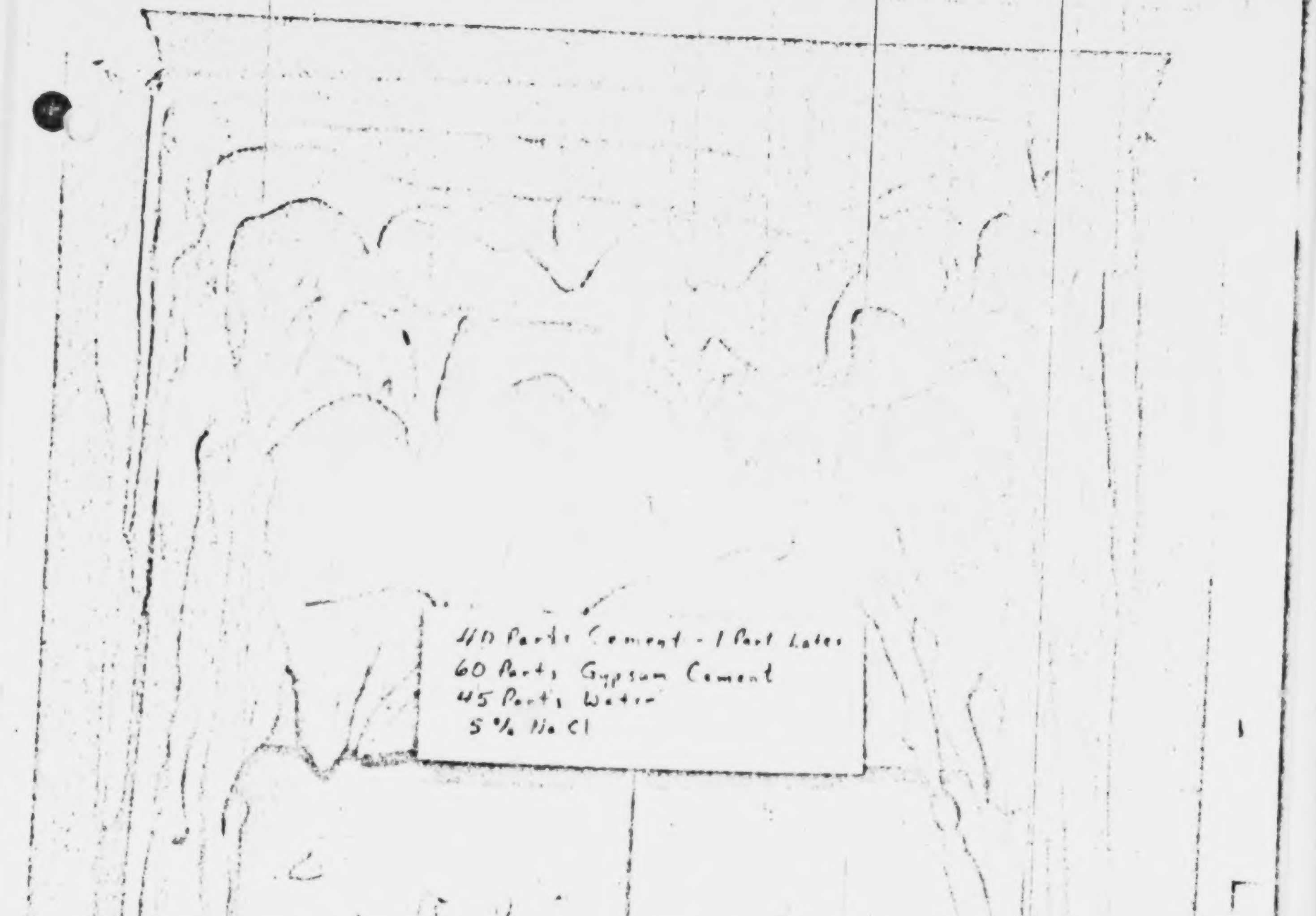
40 Parts Cement - 1 Part Latex
60 Parts Gypsum Cement
45 Parts Water
5% NaCl

FIG. 36

then filling the remaining half from the second mix. When the samples were tested after ten minutes, the failure did not occur at the interface of the two slurries.

A silicate cement grout slurry was also tested. Laboratory tests indicate a fluid time of 20 seconds. The grout was a mixture of two slurries in equal volumes. The cement portion of the slurry consisted of 50 parts Type I Cement, 50 parts Fly Ash and 40 parts water by weight. The silicate portion consisted of 75 parts 40-Baumé Sodium Silicate and 25 parts water by weight.

The two slurries were mixed separately, then pumped with centrifugal pumps through a one-inch hose to the spraying apparatus. During the initial tests, the slurry was sprayed on the walls of the full-size simulated mine tunnel, using a hand lance with a spray nozzle of 1/2-inch pipe 26 inches in length. A photograph of this operation is shown in Figure 37. It can be seen that some build-up is occurring on the floor and walls.

Further testing was done with a mechanical spraying device as shown in Figure 38. This device consisted of a rotating spray arm mounted on the end of the discharge manifold. The spray arm rotated due to the thrust created by the slurry discharging from the flattened exit nozzle. The arm was set on a pedestal 3 feet above the floor. As the slurry was pumped through the spray arm, the rotation caused the slurry to be thrown in a circular pattern around the floor, walls and roof. The material

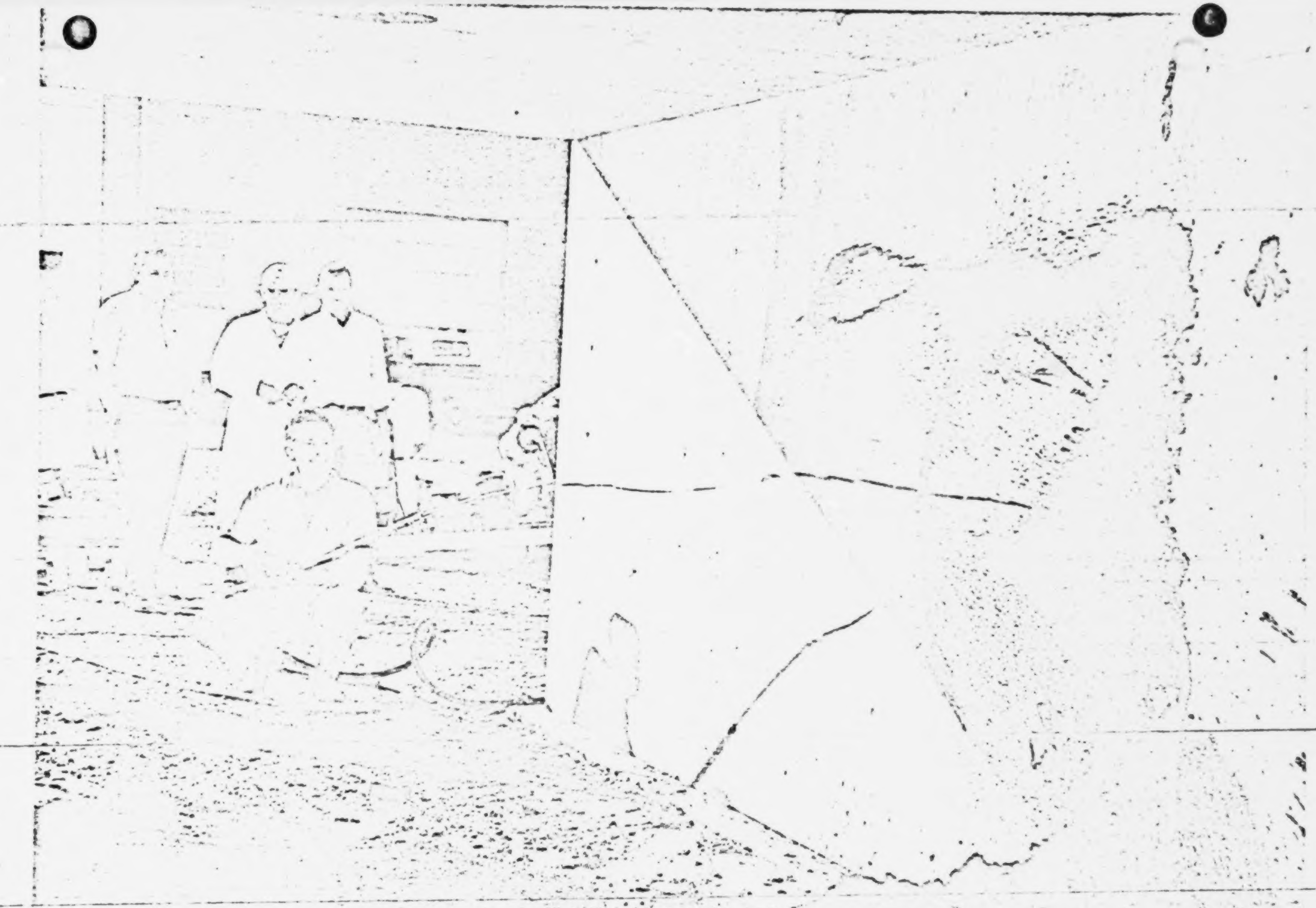

FIG. 37

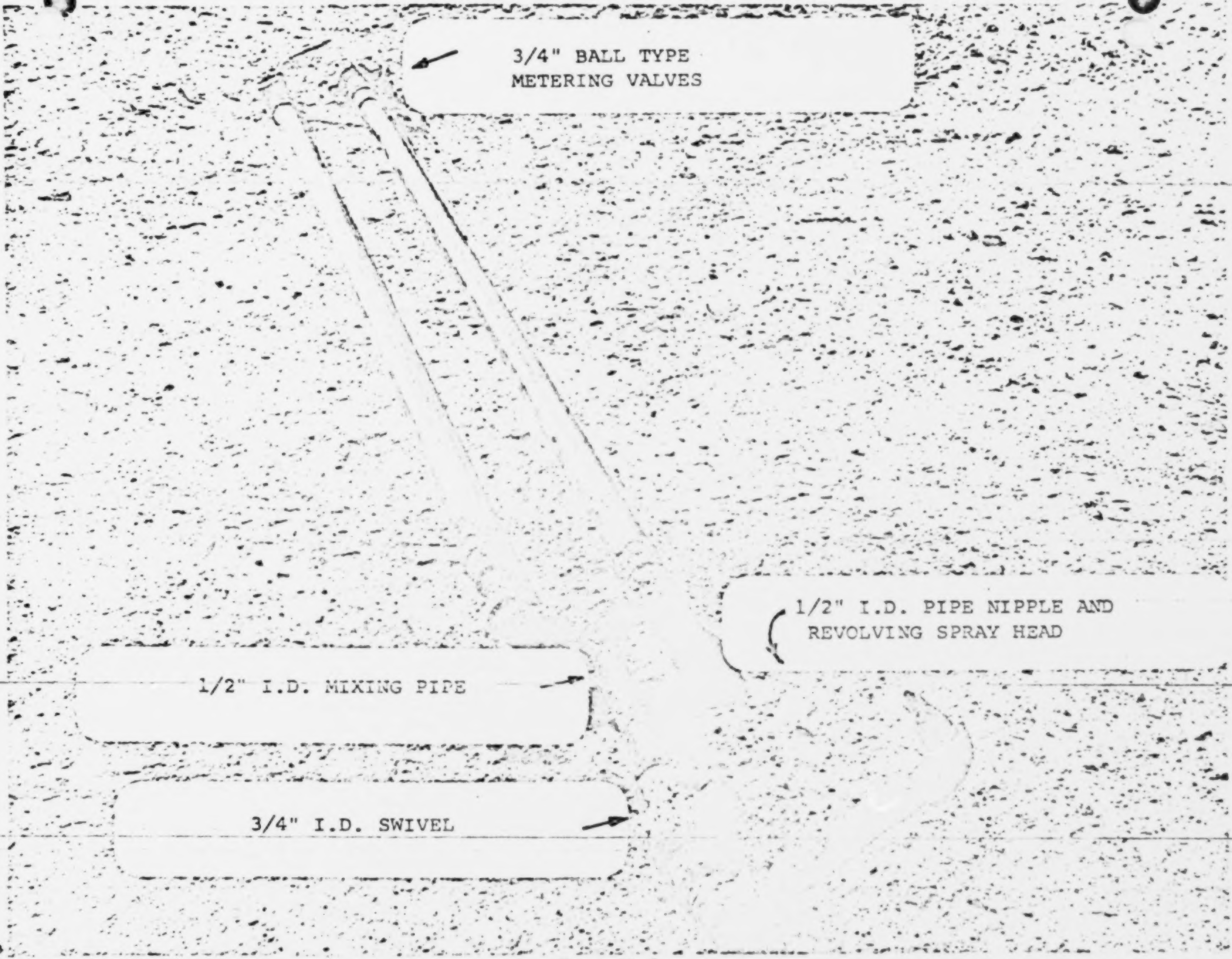
3/4" BALL TYPE
METERING VALVES
1/2" I.D. PIPE NIPPLE AND
REVOLVING SPRAY HEAD
1/2" I.D. MIXING PIPE
3/4" I.D. SWIVEL

Fig. 38

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would build up to about 6 inches thick on the roof, then large pieces would drop off under their own weight.

The cost of a bulkhead in a 7 by 12 foot drift using the latex-cement grout would be approximately $130.00 per cubic yard. The cost using the silicate-cement grout would be about $18.00 per cubic yard. A bulkhead with a minimum thickness of one foot would require 3 to 4 cubic yards.

This method definitely seems to have possibilities, but before the technique can be applied in a field test, additional laboratory testing should be pursued.

E. Testing Connected with Field Program

In this phase of the experimental test program, tests were conducted at the Duncan, Oklahoma facilities to confirm the feasibility of placing an aggregate material by pneumatic means into a mine drift opening to effect a bulkhead. Later tests were made in Weston, West Virginia with the equipment which would be used to place the material in a selected area mine drift.

(1) Pneumatic Placement of Material

Tests were conducted in Duncan, Oklahoma to confirm the feasibility of pneumatically conveying 3/4-inch graded aggregate and 3/8-inch graded aggregate. The following equipment was used in these tests:

1 - 200 Cu. Ft. Pressure Tank, Trailer Mounted

1 - 400 Cu. Ft. Pressure Tank, Buried.

1 - 50S-1 Compressor, LeRoi, 465 SCFM @ 40 psi, 75 HP.

1 - 25' x 5-1/2" I.D. Carbolized Hose connected to discharge on the tank, 100 feet steel pipe, victaulic coupled, 25' x 5-1/2" Carbolized hose connected to fill on tank. (Total length - 150 feet)

All volumes of air are expressed as SCFM, or standard cubic feet per minute. This is defined as the volume of one cubic foot, measured at 14.73 pounds per square inch absolute and 60° F. Compressor ratings are shown as the volume at the low pressure orifice, or intake, and the pressure at the discharge outlet.

The tests were set up to convey 3/4-inch graded aggregate and 3/8-inch graded aggregate from a 200-cubic foot pressure tank through 50 feet of 5-1/2-inch hose and 100 feet of 5-inch pipe to a 440-cubic foot pressure tank. For each test, 13,800 pounds of each aggregate size was used. A schematic drawing of the test site is attached as Figure 39.

Each material was blown from tank to tank 12 or more times with discharge time recorded.

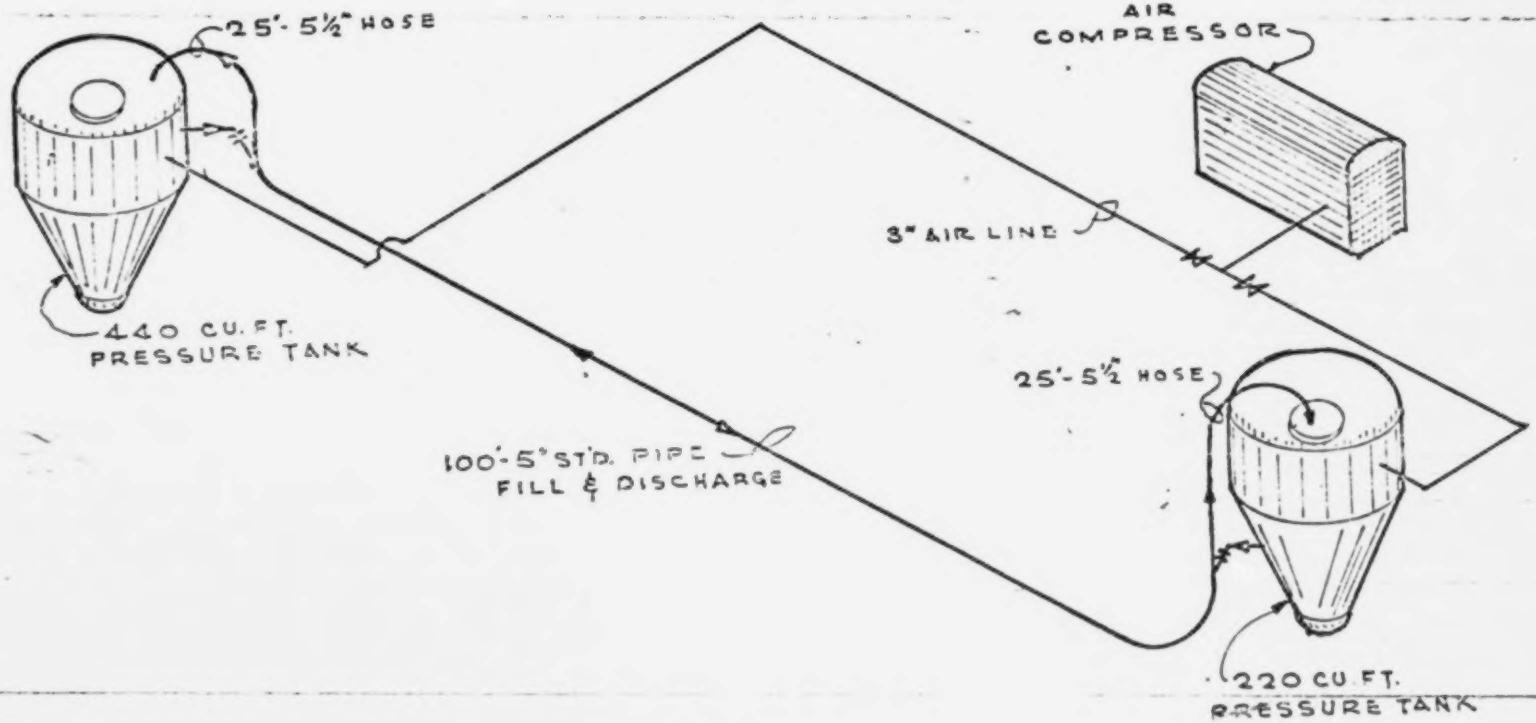
The results of these tests showed that the following rates could be expected or exceeded:

> Graded 3/4-inch aggregate could be conveyed at a rate of approximately 600 pounds per minute at an average pressure of 20 psi
>
> Graded 3/8-inch aggregate could be conveyed at a rate of 627 pounds per minute at an average pressure of 20 psi

From the results of these tests it was concluded that the 3/8-inch graded aggregate could be conveyed at a faster rate than the 3/4-inch graded aggregate, provided that the material was dry and other conditions were equal. It was also confirmed that the process was feasible and further testing was justified, but should be done with the available equipment in West Virginia which would be used in a field application.

Tests were conducted later at Weston, West Virginia. The following equipment was used in these tests:

1 - Body Load Pneumatic Bulk Transport with two 150 cubic foot pressure tanks and 175 SCFM @ 45 psi compressor


25'-5½" HOSE
AIR COMPRESSOR
440 CU. FT. PRESSURE TANK
3" AIR LINE
25'-5½ HOSE
100'-5" STD. PIPE FILL & DISCHARGE
220 CU. FT. PRESSURE TANK

TEST SITE
PNEUMATIC PLACEMENT TEST

FIGURE 39

1 - Belt Conveyor for loading aggregate into
    pressure tank on truck

1 - Front End Loader for moving material from
    stock pile to conveyor

It was determined that 3/4-inch graded aggregate and 3/8-inch graded aggregate could be conveyed through 20 feet of 4-inch hose and 60 feet of 4-inch pipe with available equipment by loading the rear pressure tank with 150 cubic feet of aggregate and using the front pressure tank for reserve air storage. However, due to the volume of air required, the truck compressor was not sufficient to discharge the complete tank of aggregate continuously, so test runs were not of sufficient duration to establish a rate of discharge. Tests did indicate that the equipment could be used satisfactorily even though rates of discharge had not been established. It was concluded that this equipment could be used under actual conditions at the mine site with the addition of a compressor capable of delivering 300 SCFM at 40 psi to insure a steady flow of aggregate.

An investigation was conducted on sample aggregate from two quarries in West Virginia to determine the voids present in a given quantity of the aggregate. Samples from Harrold's Limestone Quarry near Wolf Summit, West Virginia were 3/8-inch graded and 3/4-inch graded limestone. Samples from Greer Limestone Company, Morgantown, West Virginia, were not marked but appeared to be minus 1/4-inch and 3/8-inch to 1 inch.

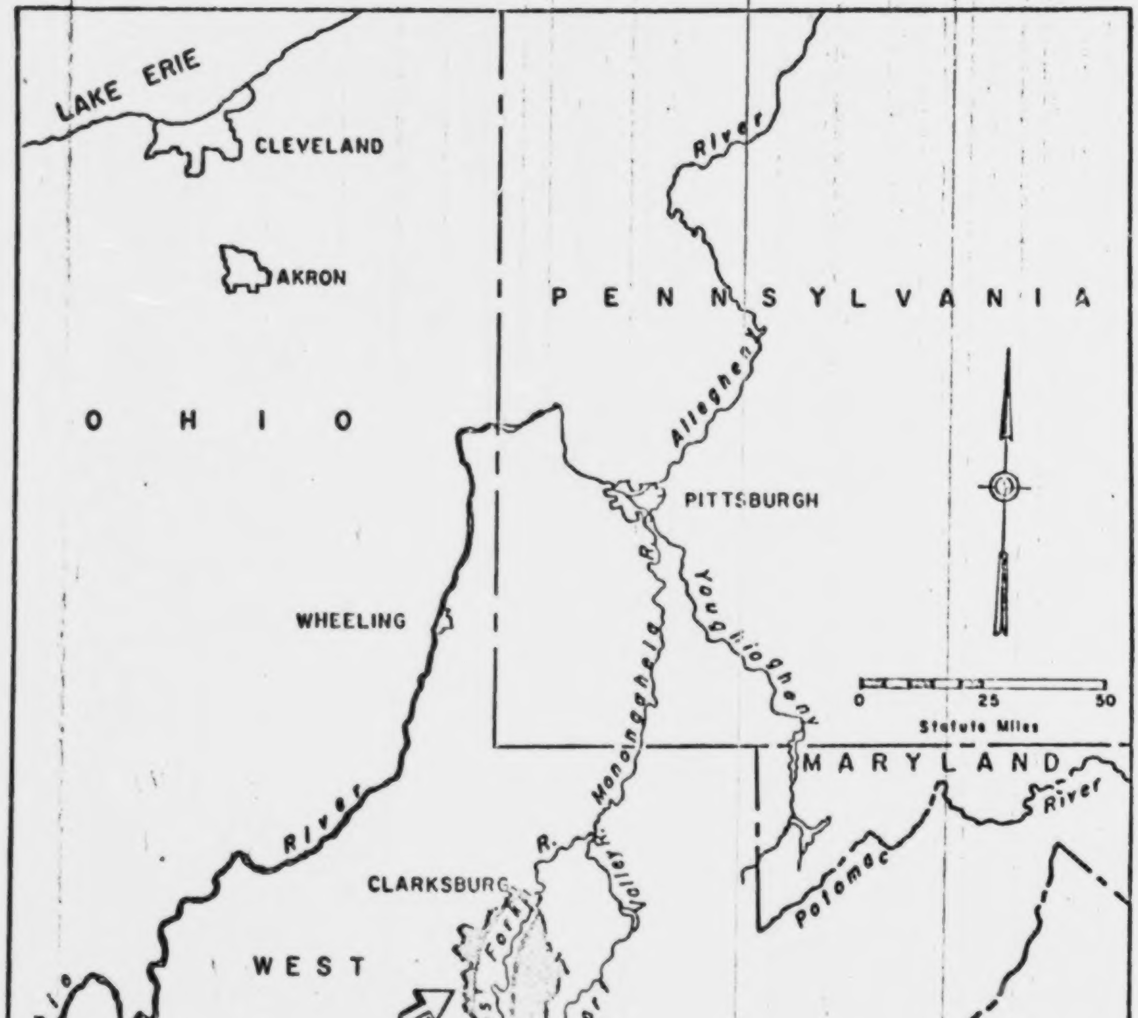
Results, shown in Table X, indicated an average of 45% voids for the 1/4-inch and 3/8-inch graded aggregate and 49% for the 3/4-inch graded. The dry density of the loose aggregate averaged 99 pounds per cubic foot for the 3/8-inch graded and 95 pounds per cubic foot for the 3/4-inch graded.

TABLE X

GRADED LIMESTONE SAMPLES - WEST VIRGINIA

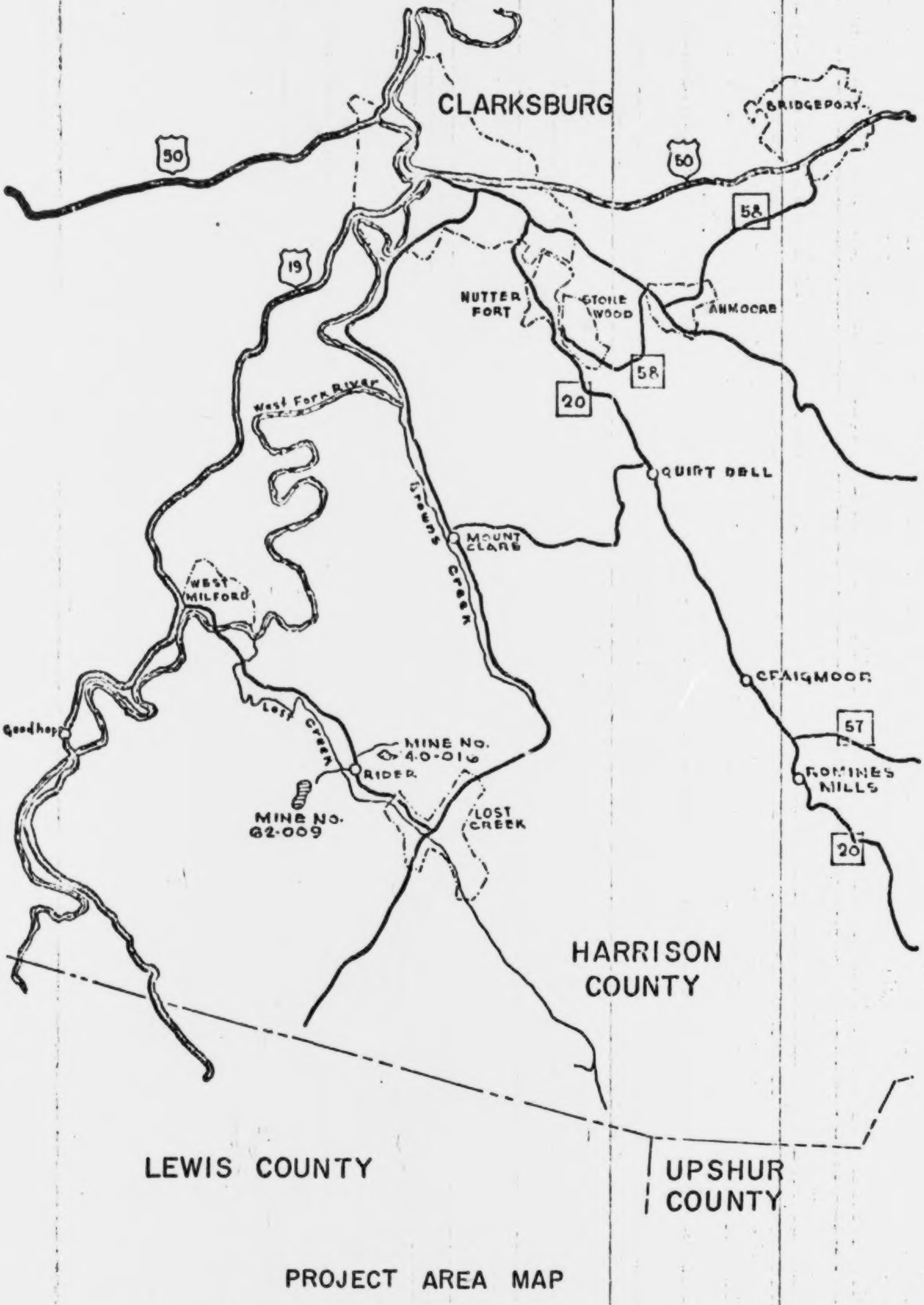
		Harrolds		Greer	
		3/8" Graded	3/4" Graded	Minus 3/4"	3/8" to 1"
Specific Gravity		2.74	2.71	2.72	2.70
Bulk Density, Lbs./ Cu. Ft.	Loose	85.77	78.50	87.60	84.60
	Packed	98.78	93.16	100.20	97.00
Gallons Limestone per 100 lbs.-Absolute		4.40	4.43	4.41	4.45
Gallons Limestone per 100 lbs.-Bulk	Loose	8.72	9.53	8.54	8.84
	Packed	7.57	8.03	7.47	7.71
Gallons Voids per 100 lbs. Limestone	Loose	4.32	5.10	4.13	4.39
	Packed	3.17	3.60	3.06	3.26
	Average	3.75	4.35	3.60	3.83
Per Cent Voids	Loose	49.5	53.5	48.4	49.7
	Packed	41.9	44.8	41.0	42.3
	Average	45.7	49.2	44.7	46.0
Cu. Ft. Slurry per Ton of Limestone	Average	10.16	11.63	9.63	10.24

III. FIELD CONSTRUCTION TESTS AND EVALUATION


LAKE ERIE
CLEVELAND
AKRON
OHIO
PENNSYLVANIA
River
Allegheny
PITTSBURGH
WHEELING
R.
Monongahela
Youghiogheny
0
25
50
Statute Miles
MARYLAND
Potomac
River
Ohio
River
R.
Valley
CLARKSBURG
West Fork
Tygart
WEST
UPPER WEST FORK RIVER SUB-BASIN
CHARLESTON
VIRGINIA
VIRGINIA

GENERAL LOCATION MAP

FIG. 40-A


CLARKSBURG
BRIDGEPORT
50
50
58
19
NUTTER FORT
STONE WOOD
ANMOORE
58
20
West Fork River
Browns Creek
QUIET DELL
MOUNT CLARE
WEST MILFORD
CRAIGMOOR
57
ROMINES MILLS
20
Goodhope
Lost Creek
MINE NO. 40-016
RIDER
MINE NO. 62-009
LOST CREEK
HARRISON COUNTY
LEWIS COUNTY
UPSHUR COUNTY

PROJECT AREA MAP

FIGURE 4D-B

III. FIELD CONSTRUCTION TESTS AND EVALUATION

Introduction

The objective of the field construction tests was to conduct additional programs for testing, evaluating and improving various methods for abating mine drainage pollution.

It was originally planned to use two mines to accomplish the desired work. Mine No. 40-016 was scheduled for testing auger hole plugs and a horizontal grout curtain. Mine No. 64-015 was to be utilized for testing new and improved methods of plugging drift mine openings. This mine was also scheduled to be flooded to provide an accelerated evaluation of the effectiveness of the mine plug, provided it is not prohibitive from a cost standpoint.

Before work could begin, it was necessary to obtain access rights to the property involved. After legal consultation, it was found that access could not be obtained to Mine No. 64-015. Other mines were then considered as a possible substitute for the mine originally selected. From those considered, Mines No. 62-008 and 62-009 were selected as a possible successor to Mine No. 64-015 with the concurrence of the Project Officer on the alternate site.

Mines No. 62-008 and 62-009 are located on Lost Creek between the towns of West Milford and Lost Creek, West Virginia. This large area of over 1,500 acres had been completely Redstone mined and contained old Pittsburgh drifts. Stripping was prevalent around its perimeter. Augering was present in both Redstone and Pittsburgh sections. This area on the northeast side of the hill is Mine No. 62-008. A new Pittsburgh drift Mine No. 62-009 had been initiated on the southeast side across the hill from Mine No. 62-008. Auger holes were located adjacent to the new workings. It appeared that this mine would be suitable for use in the project.

With the concurrence and aid of the FWPCA Project Officer, the owner of Mines No. 62-008 and 62-009 was contacted and an agreement for access rights was obtained.

In like manner, access rights were obtained for Mine No. 40-016. However, it was necessary to obtain signatures of over forty heirs to the property to obtain the rights.

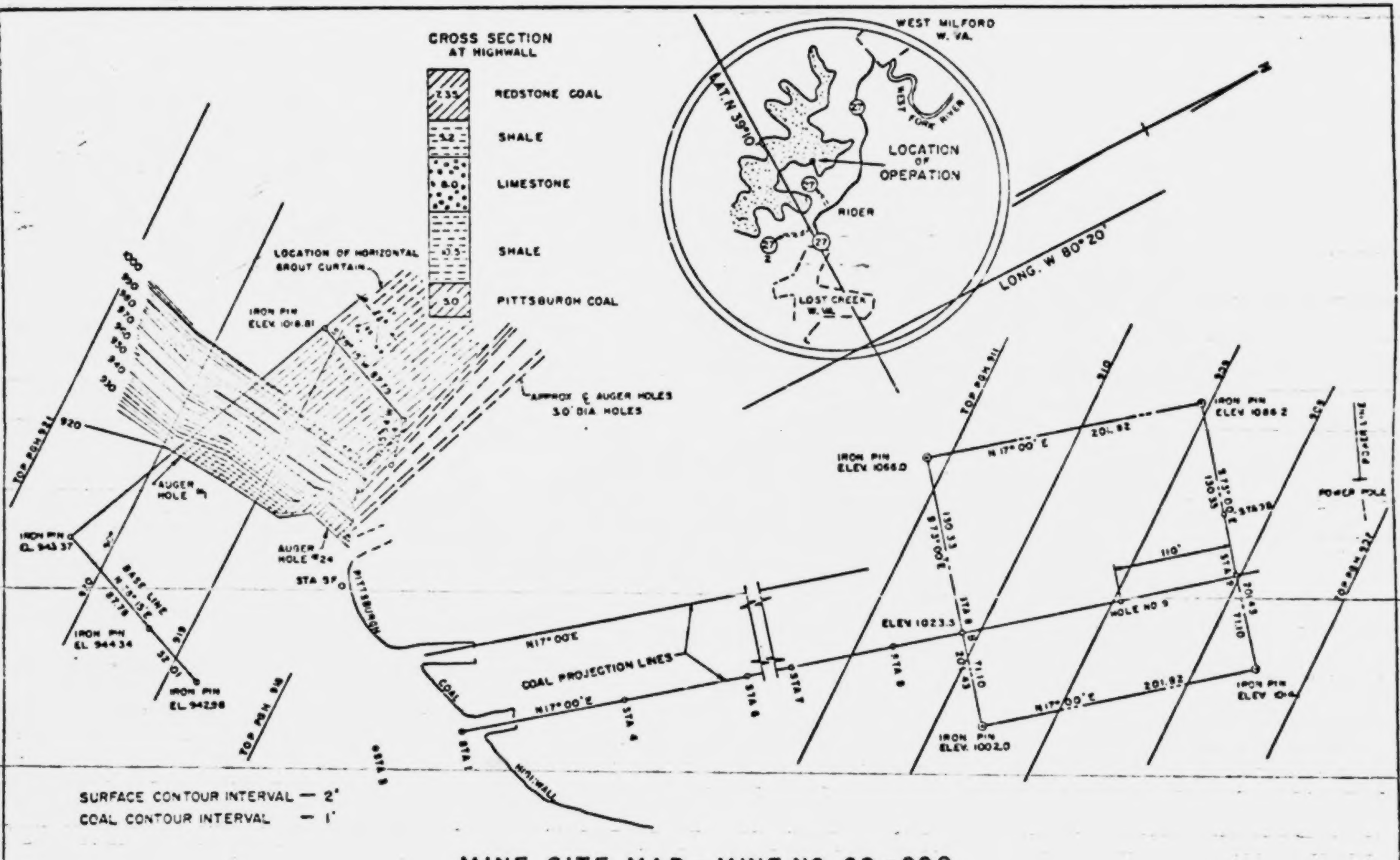
As a result of obtaining an alternate mine site, the field operations were scheduled with the concurrence of the Project Officer. The schedule included utilization of Mine No. 40-016 for hydraulic cleaning and plugging of two drift openings in the Pittsburgh seam. Mine No. 62-009 would be used for testing placement of a horizontal grout curtain and plugging of auger holes.

Seismic operations would be scheduled to attempt definition of mined areas in Mine No. 62-008 and to determine thin high wall sections in Mine No. 62-009.

In lieu of flooding a mine, the installation of a water level recorder on Mine No. 14-042A was authorized by the Project Officer.

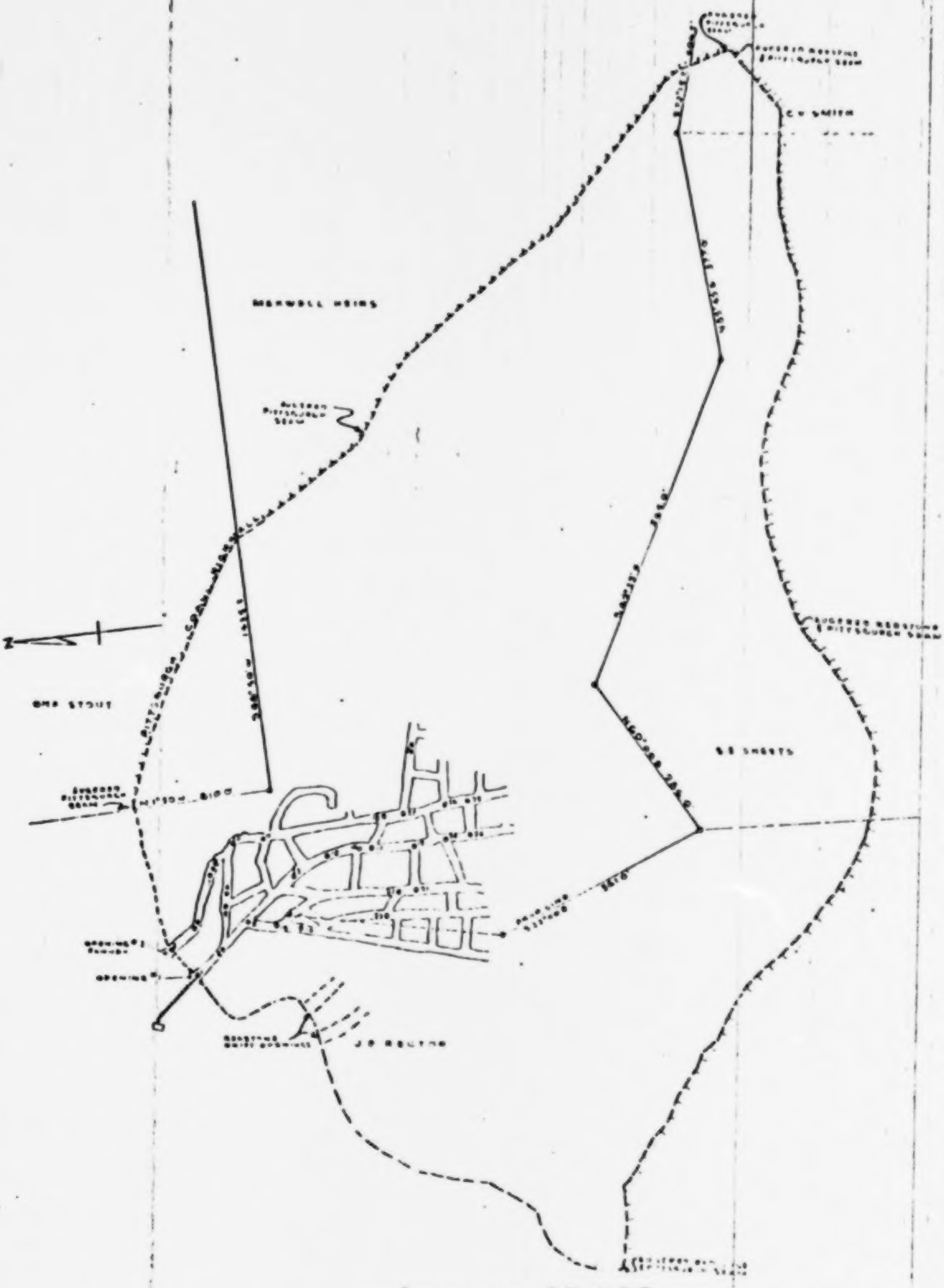
An aerial photograph showing Mines No. 62-008 and 62-009 is attached as Figure 41. Mine Site Map for Mine No. 62-009 is attached as Figure 42. An aerial photograph of Mine No. 40-016 is attached as Figure 43. This mine is located across State Road 27 east from Mine No. 62-009. Figure 44 shows a mine map of Mine No. 40-016.

FIGURE 11


CROSS SECTION AT HIGHWALL
2.35 REDSTONE COAL
5.2 SHALE
8.0 LIMESTONE
13.5 SHALE
3.0 PITTSBURGH COAL
WEST MILFORD W. VA.
WEST FORK RIVER
LOCATION OF OPERATION
RIDER
LOST CREEK W. VA.
LAT. N 39°10'
LONG. W 80°20'
N
LOCATION OF HORIZONTAL GROUT CURTAIN
APPROX. ℄ AUGER HOLES 3.0' DIA HOLES
IRON PIN ELEV 1018.81
1000
990
980
970
960
950
940
930
920
TOP PGH 921
AUGER HOLE #1
AUGER HOLE #24
IRON PIN EL 943.37
BASE LINE
87.78
IRON PIN EL 944.34
52.01
IRON PIN EL 942.98
TOP PGH 918
STA 5F
PITTSBURGH
COAL
STA 1
STA 3
HIGHWALL
N 17° 00' E
COAL PROJECTION LINES
N 17° 00' E
STA 4
STA 6
STA 7
STA 8
ELEV 1023.3
STA 8
TOP PGH 911
IRON PIN ELEV 1066.0
130.33
S 73° 00' E
N 17° 00' E
201.82
IRON PIN ELEV 1086.2
S 73° 00' E
130.33
STA 9
110'
HOLE NO 9
STA 9
201.43
71.10
201.43
71.10
N 17° 00' E
201.82
IRON PIN ELEV 1002.0
IRON PIN ELEV 1014
910
926
928
POWER POLE
TOP PGH 922
SURFACE CONTOUR INTERVAL — 2'
COAL CONTOUR INTERVAL — 1'
0
50
100 FEET
SCALE

MINE SITE MAP - MINE NO. 62-009

FIGURE 43



mine no. 40-016

Fig. 4.4

Summary

The work on the field construction part of the project was conducted in six phases. The first phase consisted of the application of a horizontal grout curtain over the augered section of Mine No. 62-009. This was accomplished by drilling eight holes in a pattern arranged so that communication of the grout material would form a curtain over an area of about 4,400 square feet. The grout was placed from seven of the drilled holes into the upper bedding plane of the Redstone Limestone formation at a depth of 84.5 feet through holes perforated in the well casing at that depth. After the curtain had been placed for some time, a core was made through the grouted section of the hole which had been left uncased to attempt to determine presence of the grout curtain.

This technique might be applicable in the prevention of ground water reaching a mined-out area if the conditions were suited to the use of this technique.

The second phase of the work involved tests to dispose of gob by mixing the gob with water and pumping the mixture into a mine. The tests indicated that this method of gob disposal was complicated and costly and very inefficient.

The third phase of the field construction work was the plugging of drift openings using a plug of aggregate or limestone chips placed remotely into the mine by pneumatic means. After the plug was placed, the center portion, containing large size aggregate with a void space of about 50%, was grouted to fill the

void space and form an impervious portion of the plug. This test proved that it was possible to build a plug in this manner. It is felt that future work of this type can be improved from both the standpoint of application techniques and cost to make this process an attractive method for plugging drift tunnel openings.

The fourth phase of the field construction involved plugging auger holes. Two types of plugs were tested. One type was a wooden wiper plug with grout material placed between the wipers. The other type of plug was a cloth retainer which, when filled with grout slurry, expanded to conform to the shape of the opening and effect the seal. This type of technique proved to be successful. However, the wooden type was difficult to fill and had to be used in selected auger holes. The cloth type plug was more applicable and would be the least expensive of the two to use in this application.

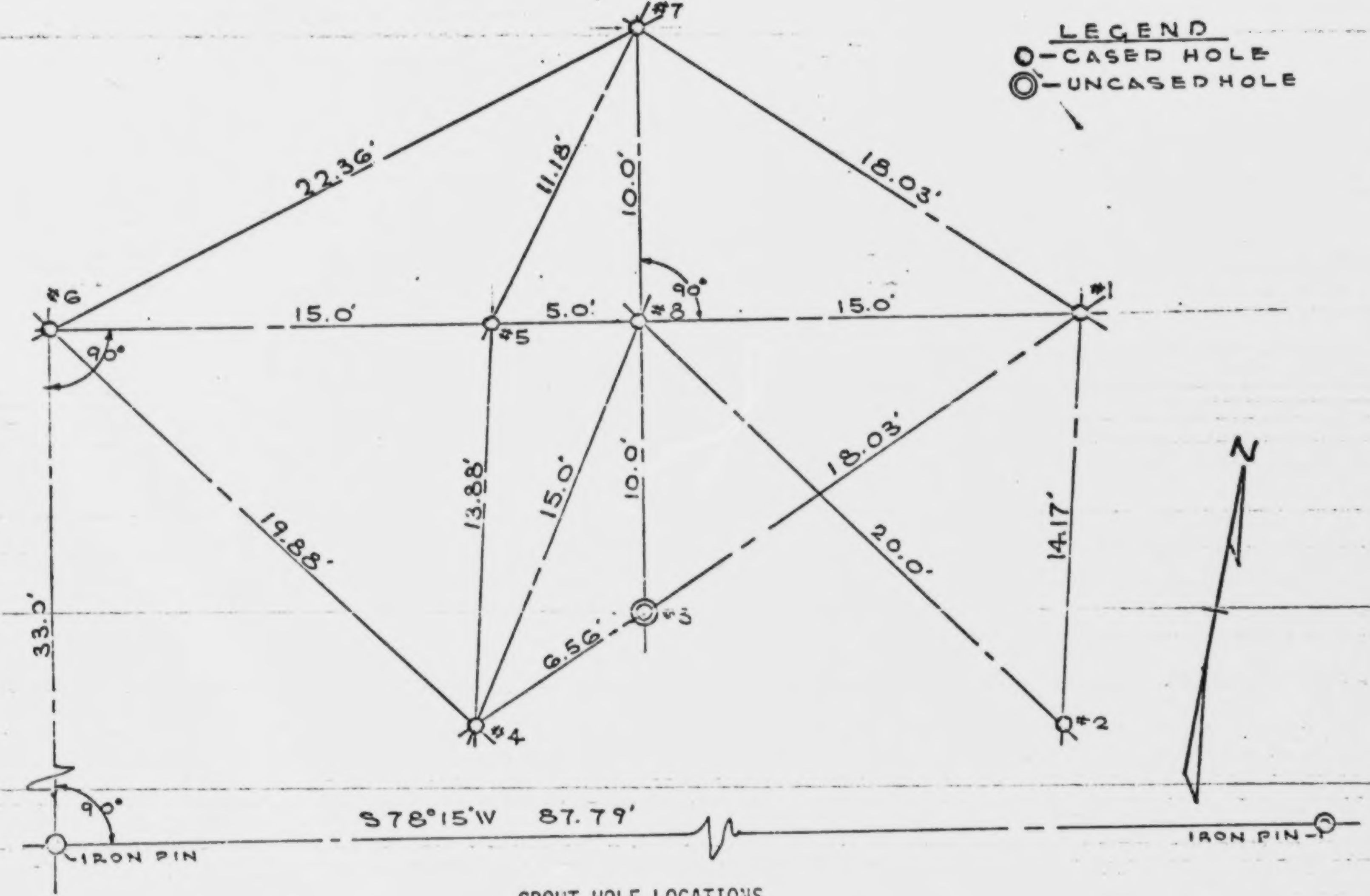
The fifth phase accomplished was the installation of a water-level recording device on Mine No. 14-042A. This was done to provide an accurate record of the conditions within the mine which had previously been plugged using Expendable Grout Retainer type seal. This installation is being monitored to provide current information and permit evaluation on the effectiveness of a mine plug. It can continue to be a source of information for the FWPCA personnel after this project has been completed.

The sixth phase was a seismic study conducted for the purpose of locating mine workings and thin high wall sections utilizing seismic techniques. The recording techniques and corrections applied were insufficient to define the mined areas. The results of the high wall tests showed reflections to be present in the mined area, but these reflections could not be distinguished from those in the unmined area. The results of the refraction tests were inconclusive.

A. Horizontal Grout Curtain

The Petitto Brothers Mine No. 2 (Mine No. 62-009) was selected for the application of a horizontal grout curtain. A horizontal grout curtain is a thin sheath of grout placed under pressure in a horizontal bedding plane to effect an impermeable curtain through which the passage of water will be greatly reduced. This area was selected to coincide with a new drift operation which had been started. It was thought that the grout curtain could be applied over an unmined area in the tunnel direction, then checked later when the mined area proceeded underneath the grout curtain, to evaluate the effectiveness of the curtain.

A one-acre site was laid out so that an extension of the new mine would pass under the area containing the grout curtain. Before work started on the one-acre area, a decision was made to test performance in a smaller area, so it was suggested by the Project Officer that we apply the grout curtain over a small area above a section of the mine which had been previously augered so the auger holes could be used for monitoring. A base line was laid out over the augered area. From the base line, the locations for 8 holes were laid out as shown on Figure 45. An access road was constructed from the Redstone coal level across the one-acre tract previously selected to the desired site above the augered section, and the site then cleared to prepare for the drilling of the holes.


LEGEND
O-CASED HOLE
◎-UNCASED HOLE
#1
#2
#3
#4
#5
#6
#7
#8
22.36'
11.18'
10.0'
18.03'
90°
15.0'
5.0'
15.0'
90°
19.88'
13.88'
15.0'
10.0'
18.03'
20.0'
14.17'
6.56'
33.0'
90°
S78°15'W 87.79'
IRON PIN
IRON PIN
N

GROUT HOLE LOCATIONS
HORIZONTAL GROUT CURTAIN
MINE NO. 62-009
FIGURE 45

Several drilling contractors were contacted and from the bids submitted, the low bidder was chosen to perform the work. Eight holes, 6-1/2 inches in diameter, were drilled in the area of the proposed grout curtain, and one hole was drilled in the original one-acre tract for use as an acoustic velocity hole for the seismic study.

Casing 2-3/8 inches in diameter was set in all of the holes except Hole No. 3. This hole was drilled 2 feet into the Redstone coal and left uncased so it could be used for coring through the grout curtain to evaluate the results of the grout curtain application.

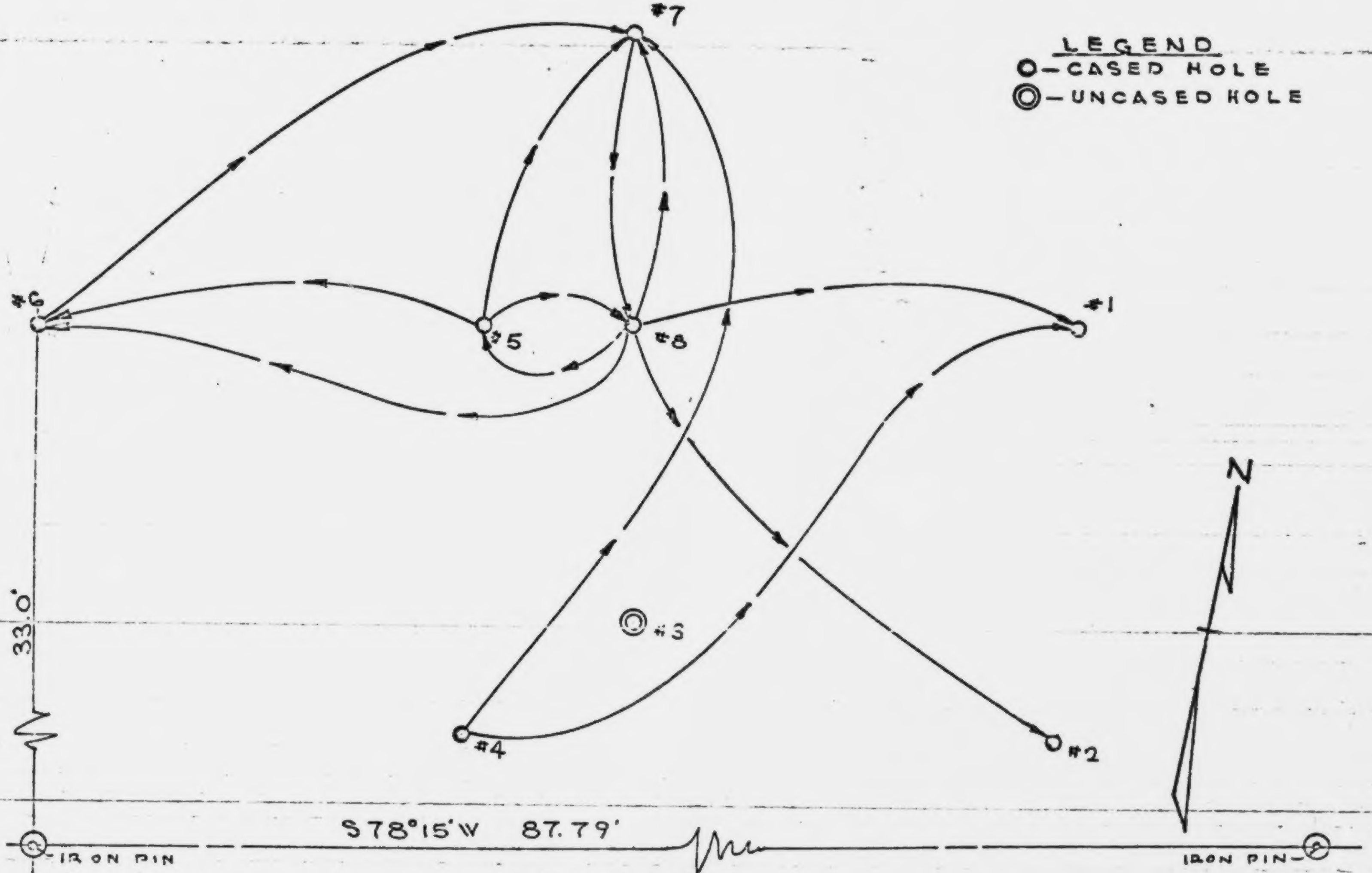
Casing was grouted in place and then perforated in the uppermost bedding plane of the Redstone Limestone formation with two 1/2-inch holes. On 2 of the wells, the perforating company was unable to lower the gun to the required depth, due to a restriction in the casing. Remedial cleaning methods permitted the perforating gun to be lowered to the required depth and the casing perforated.

Hydraulic communication tests were conducted on the completed holes. In preparation for these tests, a bubble-tube type testing apparatus was connected to each of the seven completed holes. This simple apparatus consisted of a valve on the casing from which a hose led to an adjacent container full of water. When tests were initiated, these containers were observed during the pumping operation. Fluid communication into a given hole was indicated by the presence of bubbles in the container caused by displacement of air from the hole by the infiltrating fluid.

Hydrostatic pressure was applied by a pumping unit consisting of two triplex reciprocating plunger pumps, with a 4-1/2-inch plunger having a 10-inch stroke, powered by two 335 HP turbo charged diesel engines. The capacity of this unit is approximately 750 gallons per minute at 1,000 pounds per square inch discharge pressure, with a maximum pressure capability of 10,000 psi.

Flow communication tests were made in each cased hole using water. Breakdown pressures ranged from 600 psi to 3,000 psi. Water was then injected into each hole at a rate of 400 gallons per minute with an average pressure of 350 psi to establish a pattern of communication. This flow pattern is shown in Figure 46. Tests were rerun after 24 hours to confirm results of the first tests.

The grout curtain was then placed in the bedding plane by pumping 635 gallons (85 cu. ft.) of slurry in the center hole (Hole No. 8), and 450 gallons (60 cu. ft.) in each of the remaining 6 holes, at a rate of approximately 150 gallons per minute, and displacing into the formation with water. Average pump


LEGEND
○-CASED HOLE
◎-UNCASED HOLE
#7
#6
#5
#8
#1
N
◎ #3
#4
#2
33.0'
S78°15'W 87.79'
IRON PIN
IRON PIN

FLOW COMMUNICATION
HORIZONTAL GROUT CURTAIN
MINE NO. 62-009
FIGURE 46

pressure was 100 to 300 psi. Slurry used as grout was Halliburton "Light Cement". Halliburton "Light Cement" is composed of 65 percent Cement, 35 percent Fly Ash, 6 percent Bentonite and 9.9 gallons water per cubic foot of dry blended material, or 61 parts Cement, 26 parts Fly Ash, 5.2 parts Bentonite and 82.5 parts water by weight. The data for the drilled holes and the grout curtain is shown in Table XI.

Examination of the core through the horizontal grout curtain taken in Hole No. 3 was made visually. The section containing the bedding plane where the curtain was placed is shown in Figure 47. A break in the core occurred at this point. The portion below the break is much harder than that above and seems to be free of fractures. The upper part was split in half when recovered and contained other fractures.

Evidence of cement could not be seen visually. An X-ray Diffraction test was made on the core. This test identifies compounds of cement. These compounds are calcium silicate and calcium hydroxide in amounts exceeding 5%. Results of the analysis did not indicate presence of these compounds.

TABLE XI

HORIZONTAL GROUT CURTAIN DATA
MINE NO. 62-009

	Depth From Surface-Feet					
Hole No.	Top Redstone Coal	Top Redstone Limestone	Perforated 2 Jets	Total Depth	Breakdown** Pressure-Psi	Cu.Ft. of Grout Slurry
1	69.5	82	84	94	1900	60
2	70	82	84	93	700	60
3	68.5	--	--	70.5	---	--
4	*	82	84	92.5	3000	60
5	70	82	84	93	1400	60
6	69	83	85	92.5	600	60
7	69	82.5	84.5	93	1200	60
8	70	82	84	93	650	85

Note: Casing in Holes 1,2,4,5,6,7 & 8 grouted with 10 sacks cement.

* No coal - clay vein.
** Breakdown pressure is pump pressure required to create fracture in formation.

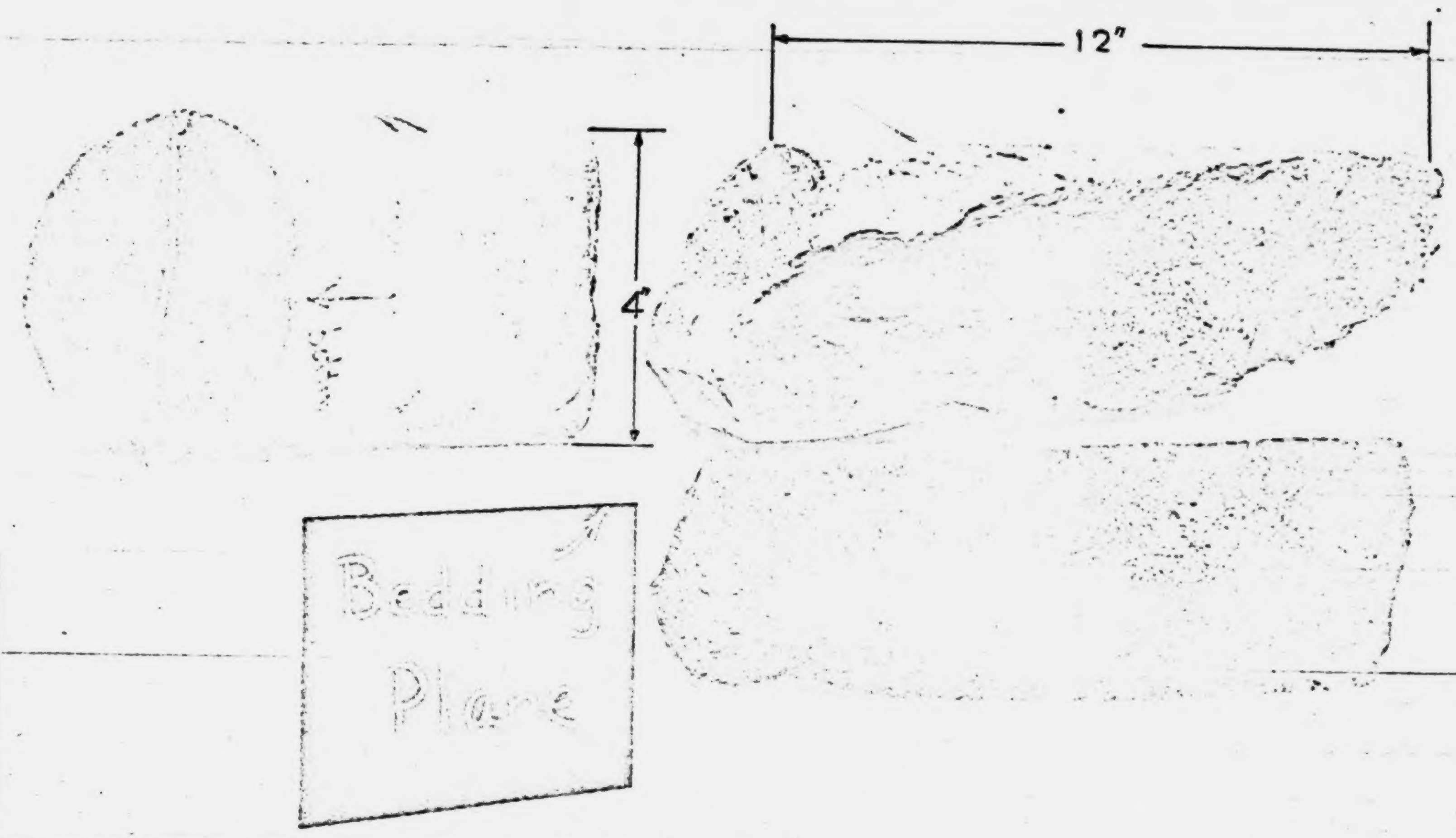

12"
4"
bottom
Bedding Plane

Fig 47

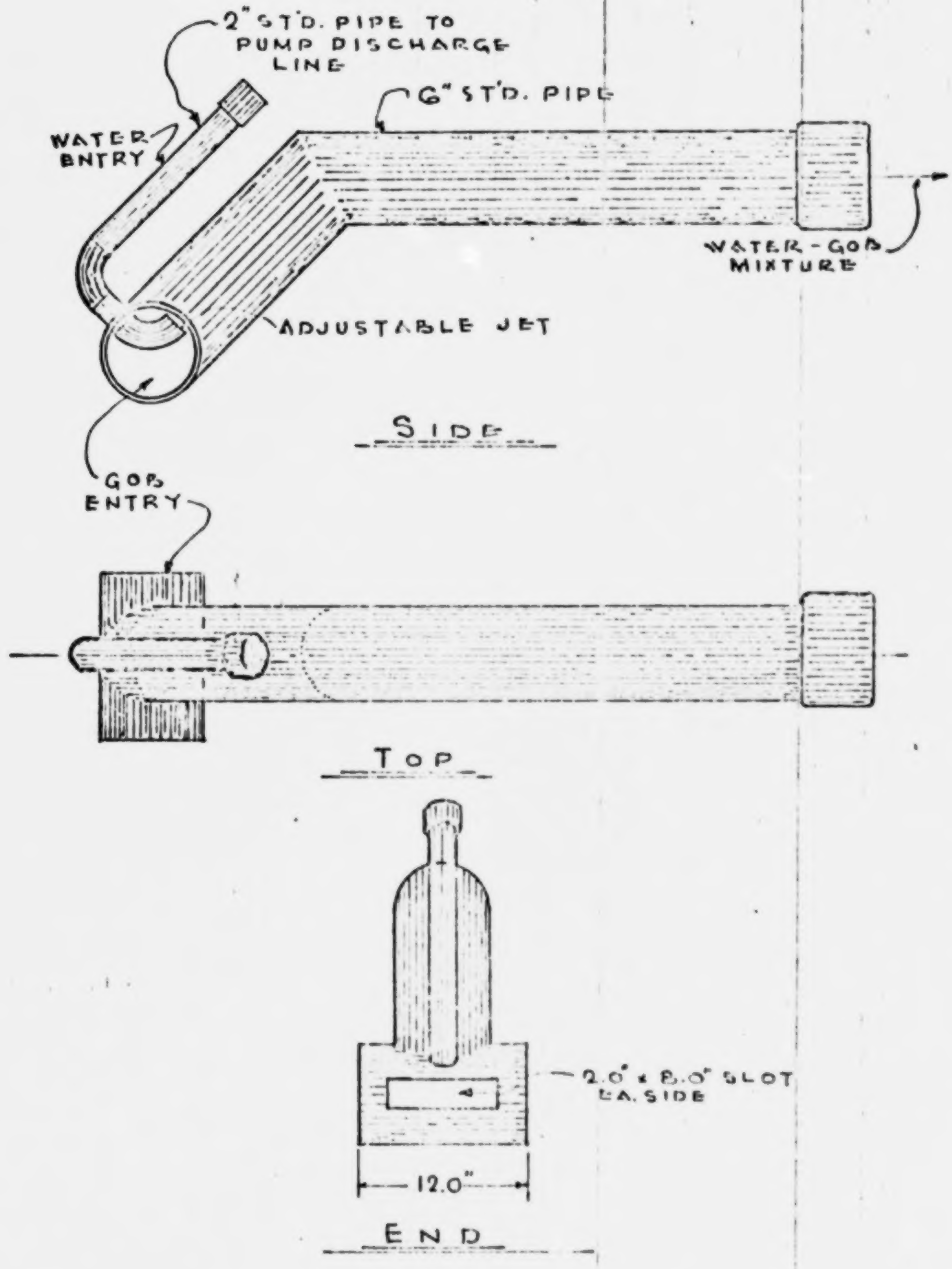
Reproduced from best available copy.

B. Gob Disposal

One phase of the field construction evaluation was a test for development of a method for disposing of gob or mine spoil. One method considered and eliminated was the movement of the gob by mechanical means into the drift tunnel. This would have entailed the use of belt conveyors or a similar system. Such a system was believed to be impractical since it would have been expensive to provide sufficient lengths of equipment to move material back into the tunnel. It would have also required considerable handling to move the gob in several areas within the tunnel. This would have been difficult to accomplish remotely.

The method chosen for field trial was the conveying of the gob in a water mixture. A test site was selected at Mine No. 62-009 in an area near the tipple. A pond for use in the test was made by erecting a small dike to contain the water. The bottom of the pond sloped so that there was a drop of about 2 feet in the length of the pond. Two sumps about 3 feet in diameter were dug to a depth of 3 feet in one corner of the pond, at the low end, to provide operational space for the pump suction and the cellar jet device. Water was then pumped into the pond for use in the test.

A cellar-jet was designed to permit the gob to be picked up and mixed with water as it was pumped through the cellar jet. The cellar jet was fabricated as shown in Figure 48.


2" ST'D. PIPE TO PUMP DISCHARGE LINE
6" ST'D. PIPE
WATER ENTRY
WATER - GOB MIXTURE
ADJUSTABLE JET
SIDE
GOB ENTRY
TOP
2.0" x 8.0" SLOT EA. SIDE
12.0"
END

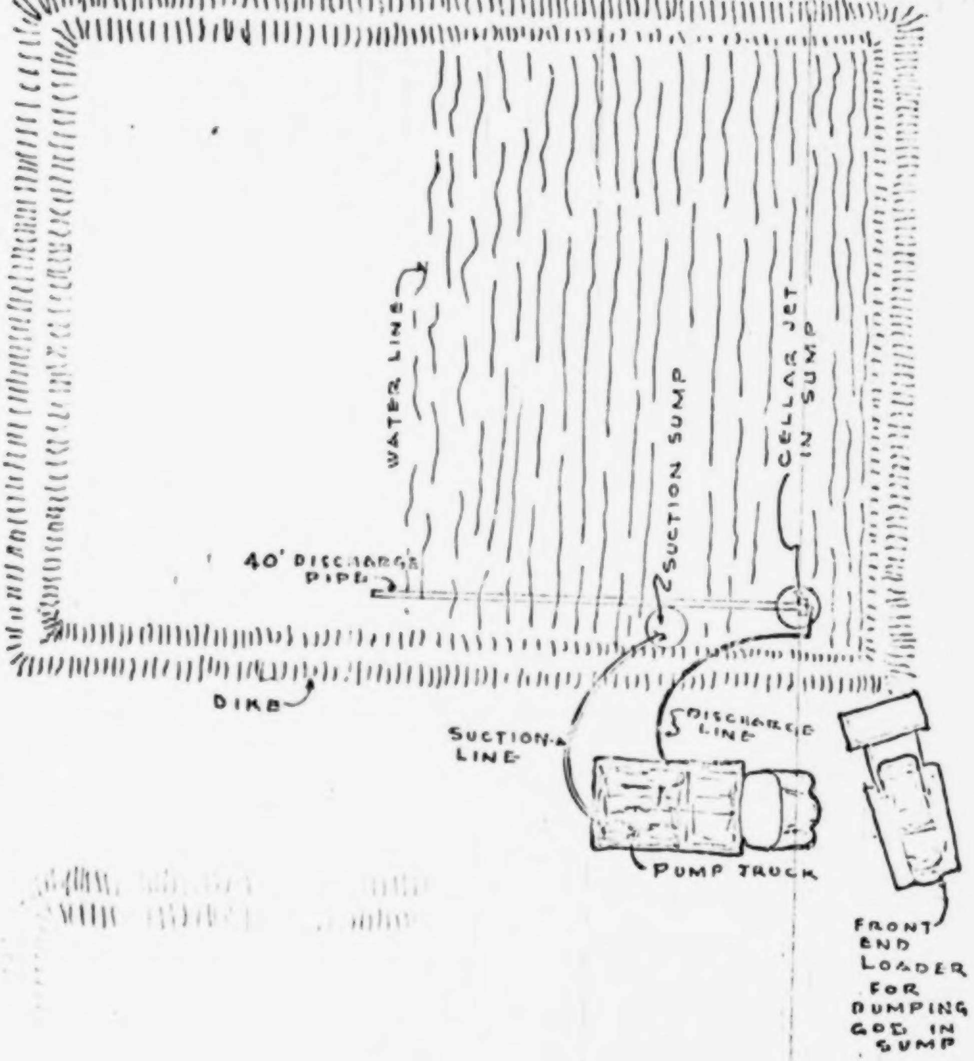
CELLAR JET FOR GOB DISPOSAL

FIGURE 48

Tests were conducted with the arrangement shown by the schematic layout in Figure 49. The procedure was to pump water through the cellar jet in the corner sump, adding the gob to the sump by means of the front end loader, and conveying the gob through 45 feet of 6-inch pipe. Material was then caught on a screen at the high end of the pond and quantity determined for each 450 gallons pumped. The pump used was a triplex reciprocating plunger pump with a 4-1/2-inch plunger having a 10-inch stroke, powered by a 335 HP turbo charged diesel engine. Pump capacity was approximately 750 gallons per minute at 1,000 psi discharge pressure. The size of the jet opening in the cellar jet could be set at 3/8", 1/2" or 2" in diameter. A maximum rate of 420 gallons per minute was obtained through the largest opening. At this rate, only 1.5 cubic feet of gob was moved per minute.

Difficulty was encountered with the size of the gob. The front end loader dumped rocks and large sections of gob in the sump, which tended to clog the cellar jet and shut off any gob from entering. The procedure was then altered. Gob was fed into the sump manually while the front end loader raised and lowered the cellar jet to prevent stoppage. Results were better but still inadequate, so the test was discontinued with the concurrence of the Project Officer.

Other factors also influenced this test. Among these were the dampness of the material, causing the clay content of


WATER LINE
SUCTION SUMP
CELLAR JET IN SUMP
40' DISCHARGE PIPE
DIKE
SUCTION LINE
DISCHARGE LINE
PUMP TRUCK
FRONT END LOADER FOR DUMPING GOB IN SUMP

TEST SITE-GOB DISPOSAL

FIGURE 49

the gob to become sticky and form large sections. This situation seemed to be aggravated when the gob was added to the water for transporting.

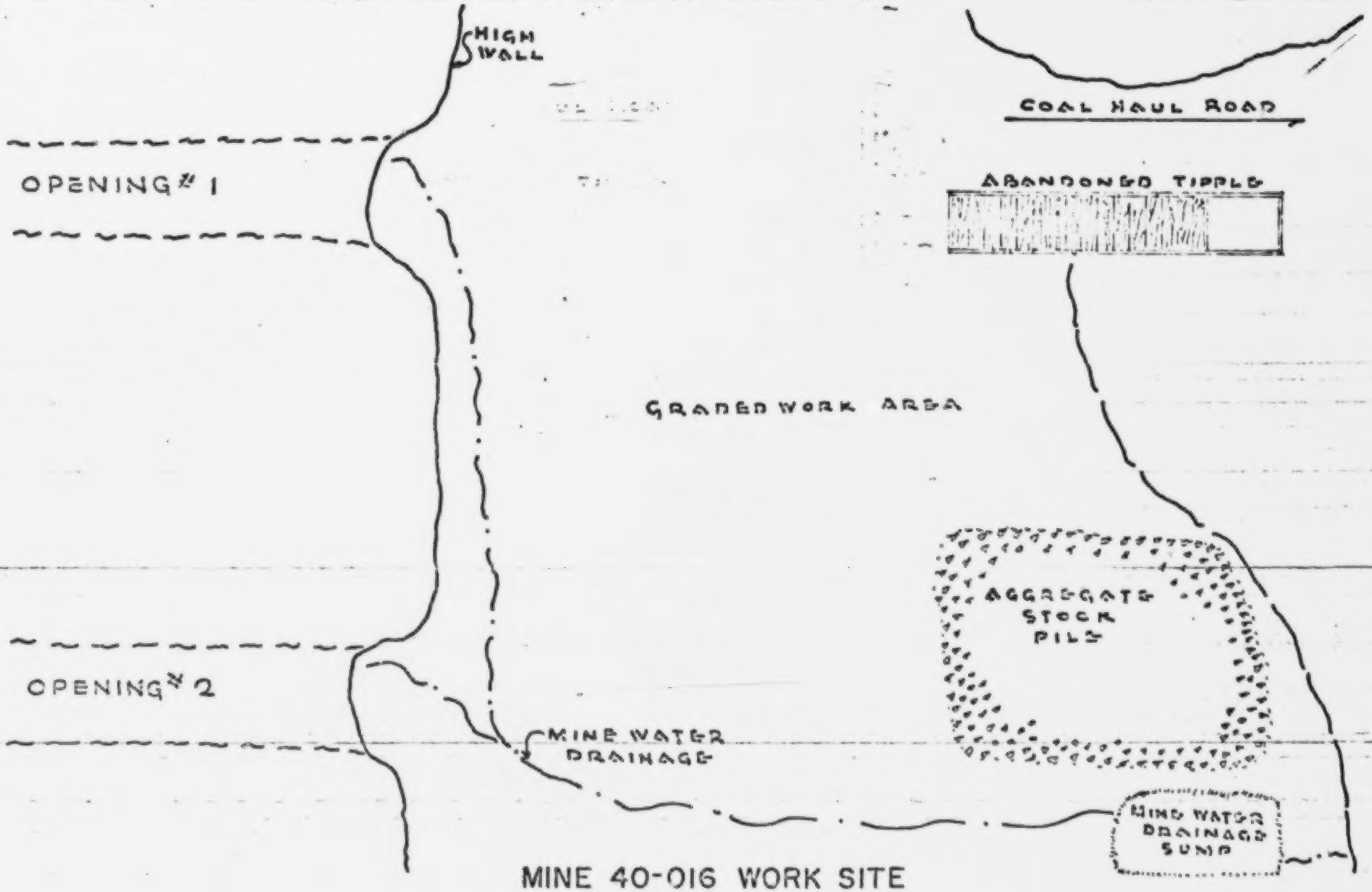
The conclusion drawn from these tests was that movement of gob into a drift tunnel for disposal can be a complicated and expensive process, since satisfactory handling of the gob would probably require crushing and grading and also perhaps a drying operation. It would also require large volumes of water to be pumped into the mine. For these reasons, it is believed to be impractical to dispose of gob in drift tunnels.

C. Plugging of Drift Openings

The plugging of the two openings in Mine No. 40-016 was originally planned to be accomplished by using Expendable Grout Retainers. A phase of the Experimental Test Program was for the improvement of these retainers. This phase is described in Part II (B) of this report. An exploration of the mine indicated that Opening No. 2 contained a row of center posts so closely spaced as to prohibit the effective placement of the retainers. The space between the center row and the side posts was 40 inches and 54 inches. The center posts were from 2 feet to 5 feet apart. The side rows of posts on both openings would have hindered an effective seal of a retainer against the wall.

At the request of the Project Officer, tests were conducted as described in Part II (E) of this report, and it was found that a plug of aggregate material of various grades could be placed remotely in the mine by pneumatically conveying the aggregate to the desired location. An impervious plug could be made by grouting this material in place. Therefore, a change was made to pursue in this phase the plugging of the openings by the use of grouted aggregate.

Mine No. 40-016 was approved for this purpose. A drawing of the work site is shown in Figure 50. Two drifts of this mine, an entrance (Opening No. 1) and a fanway (Opening No. 2), had mean dimensions of 12 feet wide and 7 feet high. Opening No. 1


HIGH WALL
COAL HAUL ROAD
ABANDONED TIPPLE
OPENING # 1
GRADED WORK AREA
AGGREGATE STOCK PILE
OPENING # 2
MINE WATER DRAINAGE
MINE WATER DRAINAGE SUMP

MINE 40-016 WORK SITE

FIGURE 50

is shown in Figure 51. Opening No. 2 is shown in Figure 52.

This operation was accomplished in three successive steps for each opening:

(a) Remote preparation of mine drift opening.

(b) Remote placement of aggregate.

(c) Grouting of aggregate plug.

(a) Remote Preparation of Mine Drift Opening

When work began on the site, the two openings were covered with sloughed material. Water impounded in the mine was seeping out under the dirt cover. The mine drifts were drained by making an opening at the water level and continuing gradually to the floor. A dozer was then employed to remove the sloughage and uncover the openings. The site was leveled and a drainage sump was dug to collect the water from the mines so it could be used in cleaning the drift. Rotten timbers were removed and new shoring installed in the front part of the drift.

Mine roof, walls and floor should be free of loose debris and thoroughly cleaned to obtain a complete bond between plug and existing surfaces. To accomplish this operation remotely, a maneuverable jet was built as shown in Figure 53A. The jet was made of 2-inch pipe mounted on a 2-wheel dolly, with a nozzle made by flattening the pipe to a narrow discharge exit of 1/4-inch. The pipe could be rotated 360°, raised or lowered for close cleaning, and moved laterally or longitudinally to permit placement anywhere

FIG. 51

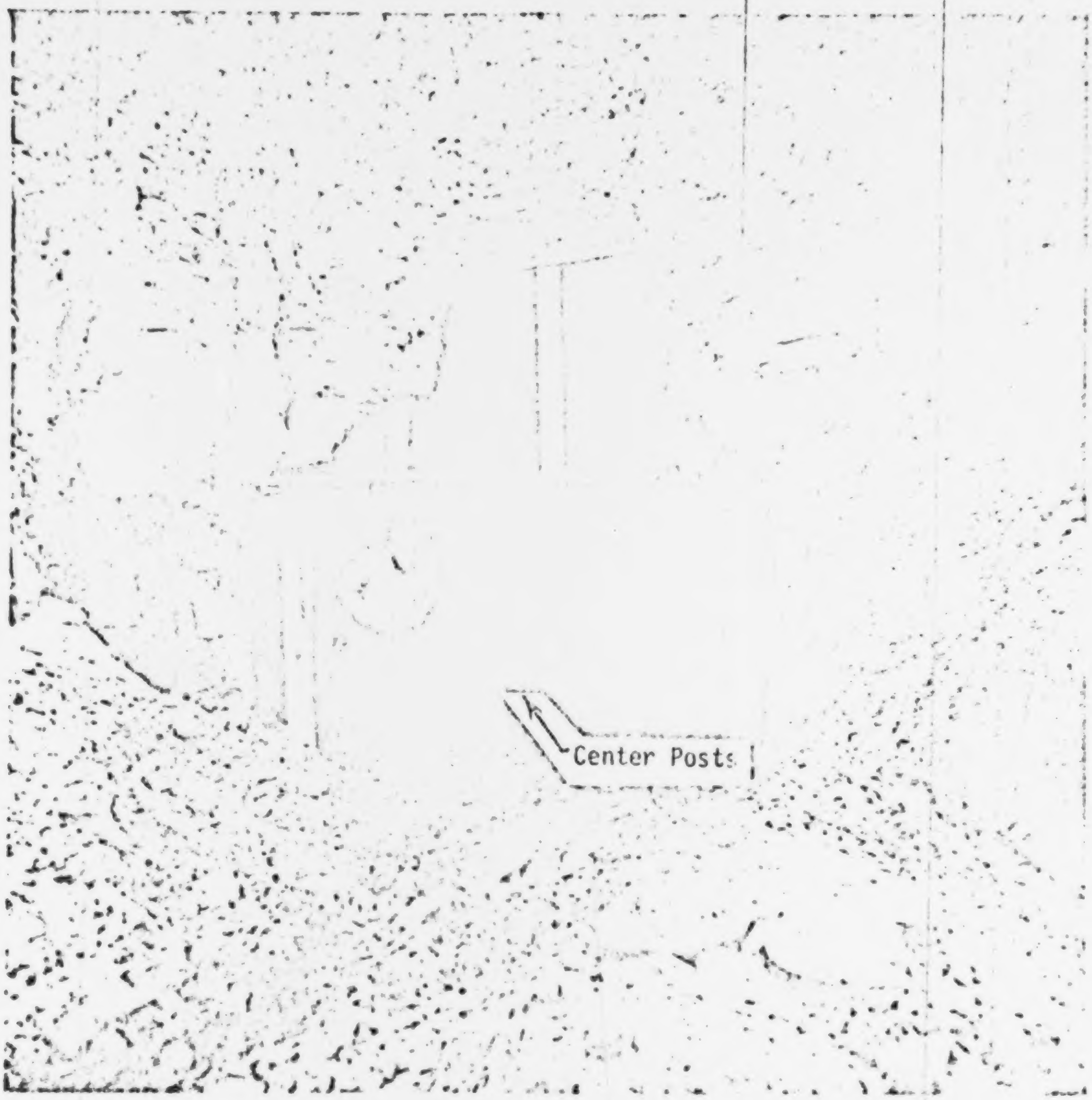
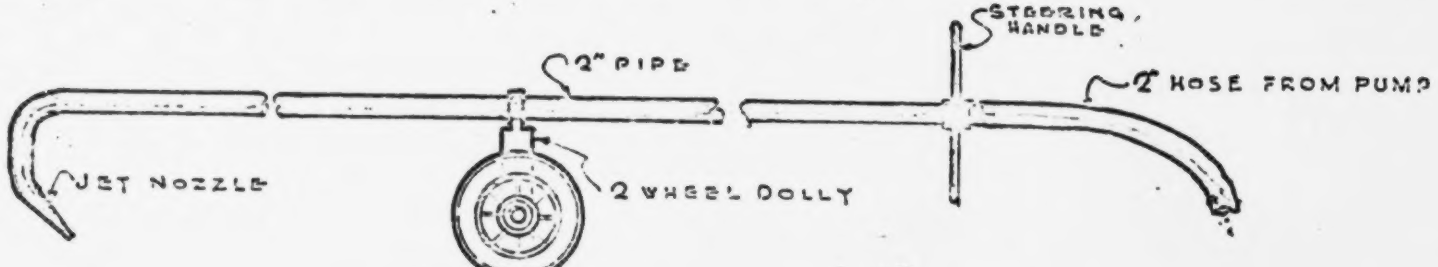
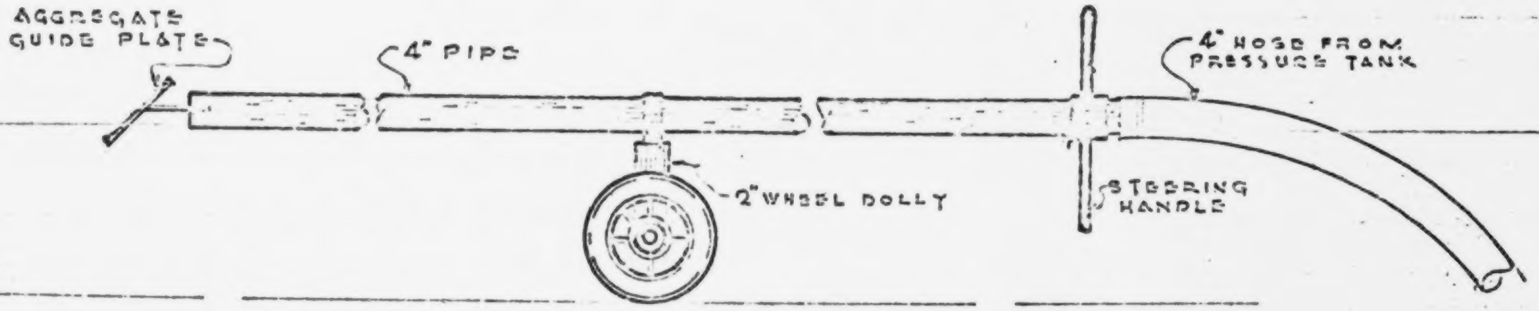

Center Posts

FIG. 52


STEERING, HANDLE
2" PIPE
2" HOSE FROM PUMP
JET NOZZLE
2 WHEEL DOLLY

(A) MANUEVERABLE CLEANING JET


AGGREGATE GUIDE PLATE
4" PIPE
4" HOSE FROM PRESSURE TANK
2" WHEEL DOLLY
STEERING HANDLE

(B) AGGREGATE PLACEMENT DOLLY

FIGURE 53

in the drift. A discharge line from a high pressure pump was connected by hose to the jet. Water was pumped through the nozzle at a pressure of 150-200 psi. Maneuvering was accomplished manually from outside the mine. Figure 54 shows the jet in a trial test prior to using it in the mine drift opening. Roof, walls and floor were cleaned with the jet and considerable debris washed out. Excess water was collected in the drainage sump and was recirculated through the jet. This method of cleaning proved to be very effective and safe. The operation required approximately 4 hours for the first drift opening and 2 hours for the second, due to improved efficiency and better accessibility.

(b) Remote Placement of Aggregate

The basic element in this technique is the placement of aggregate remotely by pneumatic means. This principle having been confirmed in previous tests, pneumatic equipment was mobilized in preparation for placement in the two selected drifts. A drain line of 2-inch plastic pipe was used in Opening No. 2 (fanway) extending from the face of the mine opening to approximately 4 feet back of the proposed plug location. The 2-wheel dolly used for jet cleaning was revised by replacing the 2-inch pipe with 50 feet of 4-inch Securaloy pipe as shown in Figure 53 (B). This was connected to a 150-cubic foot pressure tank with 40 feet of 4-inch rubber hose. Two 45° ells were affixed on the end of the discharge line to direct the flow of aggregate. This proved to be unsatisfactory due to plugging of flow at that point,

FIG. 54

so they were replaced with a 45° guide plate attached at the end of the line. This plate was held on an extension from the end of the line permitting the aggregate to discharge from the line before being deflected into position. The discharge line could be rotated 360°, raised and lowered, and rolled in or out of the drift opening manually from outside the tunnel for spot placement of aggregate.

On Opening No. 2, a bulkhead was built of 3/4- to 1-1/2-inch graded limestone approximately 30 feet back from the face of the opening by pneumatically placing aggregate from the pressure tank outside of mine through the 4-inch line to the desired area. To initiate movement of the aggregate, a surge of air at a pressure of 35 to 40 psi was turned into the 150-cubic foot pressure tank containing the aggregate. As movement of the aggregate took place, the pressure was reduced to 10 psi for the 3/4- to 1-1/2-inch size or 5 psi for the minus 3/8-inch size. A 600 cfm compressor was used in this work, but it is calculated that a 400 cfm compressor would be ample to satisfactorily perform this work in the future.

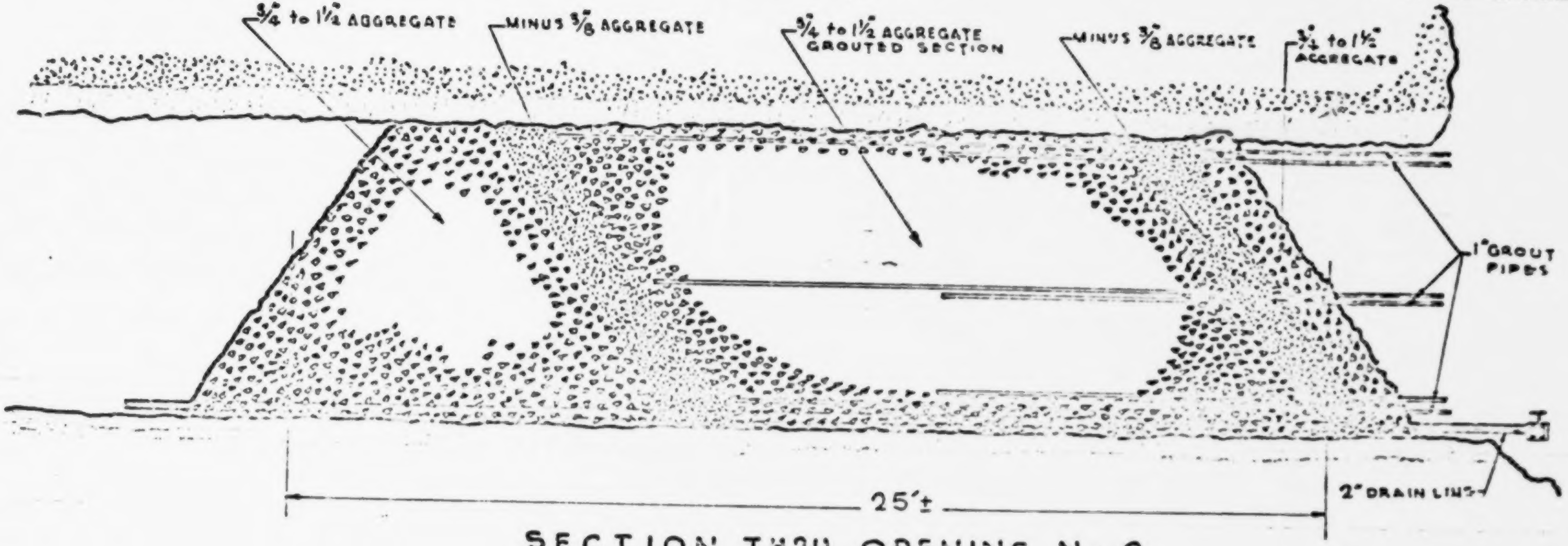
The bulkhead was approximately 14 feet wide at the floor and 2 feet wide at the ceiling contact, extending from wall to wall and from floor to roof. Upon the face of the bulkhead, minus 3/8-inch aggregate was pneumatically placed to form a barrier for retaining the grout. A thickness of 12 inches was obtained.

Three rows, vertical and horizontal, consisting of a total of 9 one-inch pipes were placed from front of mine to front of rear bulkhead for grout placement as center grouted section was filled with aggregate. This arrangement is shown in Figure 55.

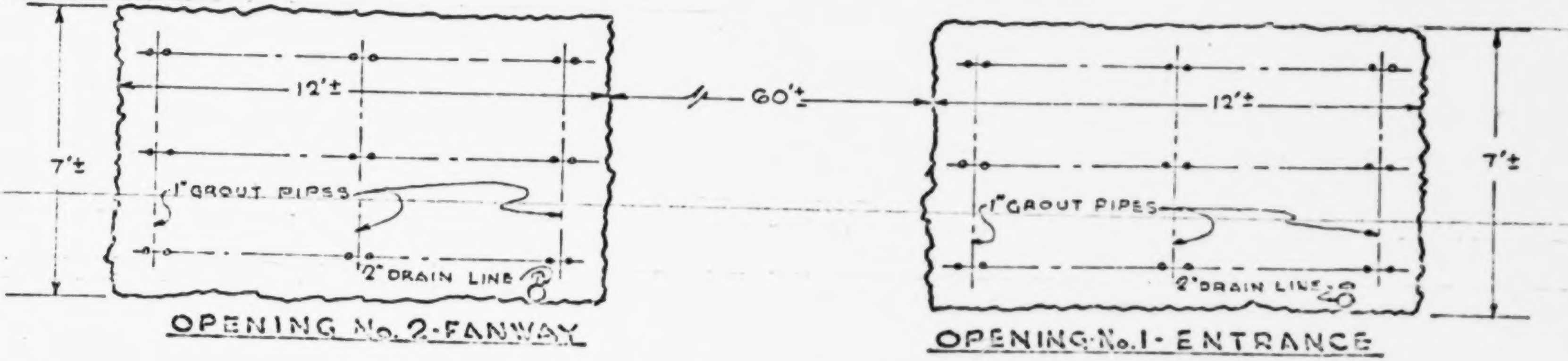
Aggregate of 3/4- to 1-1/2-inch size was then placed in the grouted section to within 12 feet of the mine face. Nine short grout pipes were placed on the same pattern as the long pipes as the aggregate was placed. Placing of this aggregate was continued to the required dimension. A layer of about 12 inches of minus 3/8-inch aggregate was placed over this to form an impermeable barrier. Twelve inches of 3/4- to 1-1/2-inch aggregate was placed over this to form a protective cover and bring the plug to its length of 34 feet. A section through the finished plug is also shown in Figure 55.

During this operation it was established that aggregate could be discharged into the mine at a rate of 1,050 pounds per minute. The tank could be loaded at 1,000 pounds per minute, using a portable belt conveyor and front-end loader. An overall average of 15 tons of aggregate per hour was placed in the mine tunnel.

The above procedure was then followed for Mine Opening No. 1. However, due to the wetting of the stockpile of minus 3/8-inch aggregate and different travel velocities caused by heterogeneous sizes of material, it became very difficult to move this aggregate from the pressure tank. This factor caused


3/4" to 1 1/2" AGGREGATE
MINUS 3/8" AGGREGATE
3/4" to 1 1/2" AGGREGATE GROUTED SECTION
MINUS 3/8" AGGREGATE
3/4" to 1 1/2" AGGREGATE
1" GROUT PIPES
2" DRAIN LINE
25'±

SECTION THRU OPENING No. 2
AGGREGATE TYPE PLUG


12'±
7'±
1" GROUT PIPES
2" DRAIN LINE
60'±
12'±
7'±
1" GROUT PIPES
2" DRAIN LINE

OPENING No. 2-FANWAY

OPENING No. 1- ENTRANCE

AGGREGATE PLUG DETAIL-MINE 40-016
FIGURE 55

a placement of only a thin layer of minus 3/8-inch aggregate and dry cement in the rear bulkhead.

(c) Grouting of Aggregate Plug

Inspection of the aggregate prior to grouting showed that while flow was continuing to come out through the bulkhead, the major portion of the flow was discharging through the drain pipe installed under the aggregate. Sufficient grout slurry would be injected to fill the void spaces in the aggregate fill. Prior tests had indicated that voids would be approximately 50% of the total space occupied by the aggregate fill.

The grout slurry selected for use in this application was 50/50 Pozzolan Cement with 2% Calcium Chloride, 18% Salt and 5.05 gallons of water per bulk cubic foot of dry material. (This slurry was composed of 56 parts Cement, 44 parts Fly Ash, 2 parts Calcium Chloride, 9 parts Salt and 50 parts water by weight). This is similar to Grout Slurry No. 7, with 2% Calcium Chloride added to give a faster set. Grout was pumped at approximately 10 gallons per minute, starting with the bottom row of grout pipes and proceeding upward as grout level indicated. The pump used was a reciprocating type capable of pumping 175 gallons per minute at 500 psi pressure, powered with a 140 HP engine.

Grouting was conducted alternately on Openings No. 1 and No. 2. Leakage of grout through the front and rear bulkheads was in evidence throughout the grouting. The front bulkhead was plugged externally with clay, but this method afforded only temporary relief and could not be applied to the rear bulkhead, so a flake cellophane material was added to the grout slurry to act as a bridge in the most permeable sections. One-half pound per sack of solids was added initially, then increased to one pound per sack for the balance of the grouting. This method proved to be effective on both front and rear bulkheads.

Moving from one mine opening to the other enabled the grout to become set in one part of the fill at one opening while grouting continued on the other opening. Using this operational procedure, the grout filled the void spaces in the aggregate fill until the fill would no longer accept any additional grout. Grouting pressures ranged from 5 psi to 50 psi.

Nine days were spent in the grouting of the two plugs in Mine No. 40-016. The job was performed almost entirely in inclement weather and under poor working conditions. A chart of the weather for the Clarksburg, West Virginia area for the four months of field operations is included as Exhibit A. Because of this and the experimental nature of the operation, costs incurred are in excess of what could be expected in future work of this type. Table XII shows a comparison of the actual cost for the plugging of the two mine openings, with an estimated cost for future work of the same type. Figure 56 shows a view of the

TABLE XII

COST INFORMATION
GROUTED AGGREGATE PLUG

Operation	Item	Actual Cost Feasibility Study Unit	Actual Cost Feasibility Study Total	Estimated Cost Future Construction Unit	Estimated Cost Future Construction Total
Cleaning	Equip. Rental		$267.00		$211.00
	Labor @ $5.00/hr.	24 hrs.	120.00	16 hrs.	80.00
	Total		$387.00		$291.00
Aggregate Placement	Equip. Rental*		$3,060.00		$ 860.00
	Material @ $3.30/ ton	300 tons	990.00	250 tons	825.00
	Labor @ $5.00/hr.	128 hrs.	640.00	32 hrs.	160.00
	Total		$4,690.00		$1,845.00
Aggregate Grouting	Equip. Rental*		$1,322.00		$ 860.00
	Material**		3,260.00		3,260.00
	Labor @ $5.00/hr.	144 hrs.	720.00	32 hrs.	160.00
	Total		$5,302.00		$4,280.00
Total Cost for Closing 2 Opening			$10,380.00		$6,416.00
Cost per Opening Closed			5,190.00		3,208.00

* Equipment Rental reduction in future construction based on renting specially designed equipment rather than using locally available construction equipment.

** Cementitious materials furnished by Halliburton Services

FIG. 56

work site after completion of the two plugs. Figure 57 shows a close-up of the completed plug in Opening No. 1. Figure 58 shows the completed plug in Opening No. 2.

It was concluded that this method is one which could be applicable in closing virtually any drift opening. In this study, Opening No. 2 contained a row of center posts which would have prevented any closure technique requiring a clear opening between the side walls for proper placement.

The grouting pipe arrangement can be improved for future efforts. The long pipes used in this work were too close to the rear bulkhead and it is believed that some washing through the finer material was experienced and caused excessive leakage. On future work, the long pipes should be plugged on the ends and side ports used through the grouted section. It is also concluded that only the top and bottom rows of grouting pipes will be necessary on future construction.

The slurry used in the aggregate grouting is satisfactory. It is basically Slurry No. 7 with 2% Calcium Chloride and one pound of cellophane flakes per sack of solids added. It could be used in further work of this type.

It should be noted that a total shut-off was not obtained in the two openings. Opening No. 1 was leaking 3.46 gallons per minute through the bottom of the plug on February 23, 1968 with a head behind the plug 14-1/2 inches above the top of

FIG. 57

FIG. 58

the protective deck, or approximately 8 feet above the floor. Opening No. 2 was leaking at the rate of 3.75 gallons per minute on February 23, 1968 with a head behind the plug of approximately 10 feet above the floor. Other leakage is occurring through the coal seams on either side of the two openings.

A more positive shut-off could be obtained with a grouting treatment using a quick-setting chemical if further reduction in flow is desired through the aggregate plug or coal seams. Several types of chemical grouting materials are available. Some of these are:

PWG - A low viscosity polymeric gel producing material.

Herculox - A hard setting resin material with high compressive strength.

Blox-All - A lignin type gel material

Formasil - A silicate material, generally used for temporary shut-offs.

Actual cost will be dependent on the volume to be used, but in small quantities, the cost should range from approximately $0.60 per gallon for Formasil to $2.05 per gallon for PWG, exclusive of placement costs.

D. Auger Hole Plugging

This phase of the field construction was to evaluate the use of closure plugs in auger holes. The site which was originally selected was about 600 feet from the new mine opening so that it would not interfere with mine work actively being pursued. But a site adjacent to the new mine was subsequently selected in a conference with the mine owner. (See Figure 42). The site after all the holes had been exposed is shown in Figure 59.

There were 30 holes exposed. The diameter varied from 38 to 39-1/2 inches horizontally and from 40 to 41 inches vertically. It was proposed to leave 6 openings nearest the mine workings to avoid interference with mining operations, and plug the other 24 openings. The holes are numbered consecutively from the hole farthest from the new mine workings back toward the workings.

The material covering some of the auger holes on the side of the half wall was removed with a dozer. Figure 60 shows a close view of the auger holes after this operation. The area was marshy due to location of impounded water nearby. A dam and storage pond were made to aid in the construction work. The water from the impoundment was tested and found to be compatable for cement and it was used to mix the cement grout slurry. Holes were monitored. Hole No. 1, farthest from the mine opening, flowed 1.36 gallons per minute. Hole No. 2, adjacent, flowed .11 gallons per minute. Other holes were damp, but did not show an active flow of water.

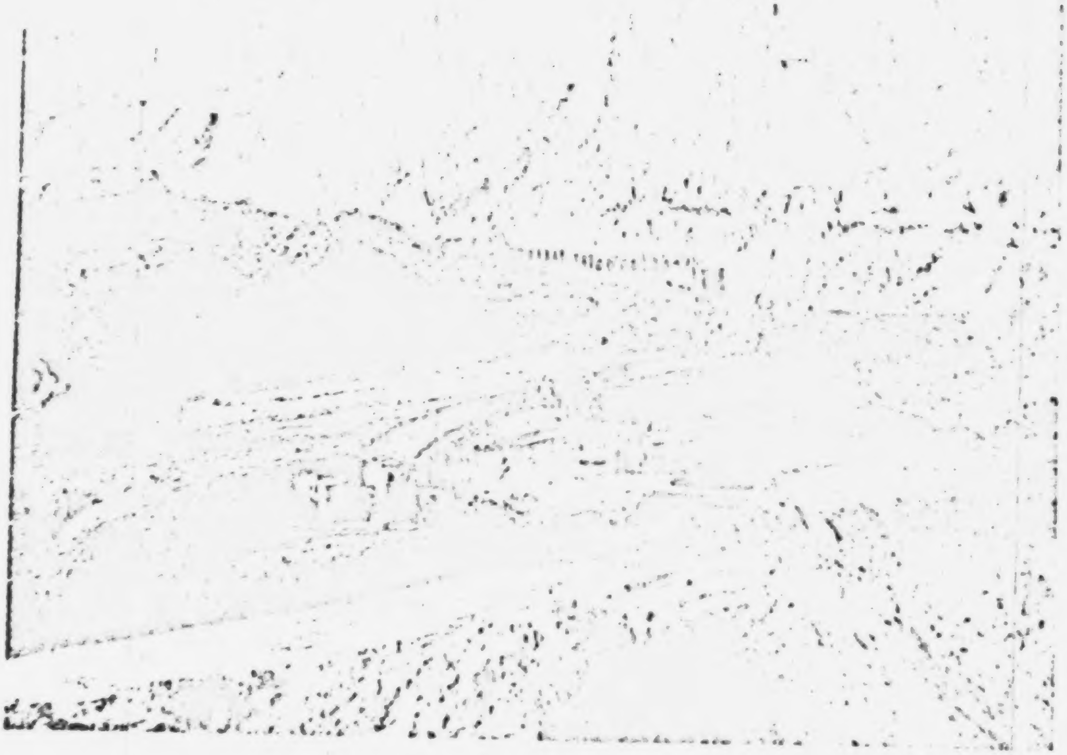


FIGURE 59

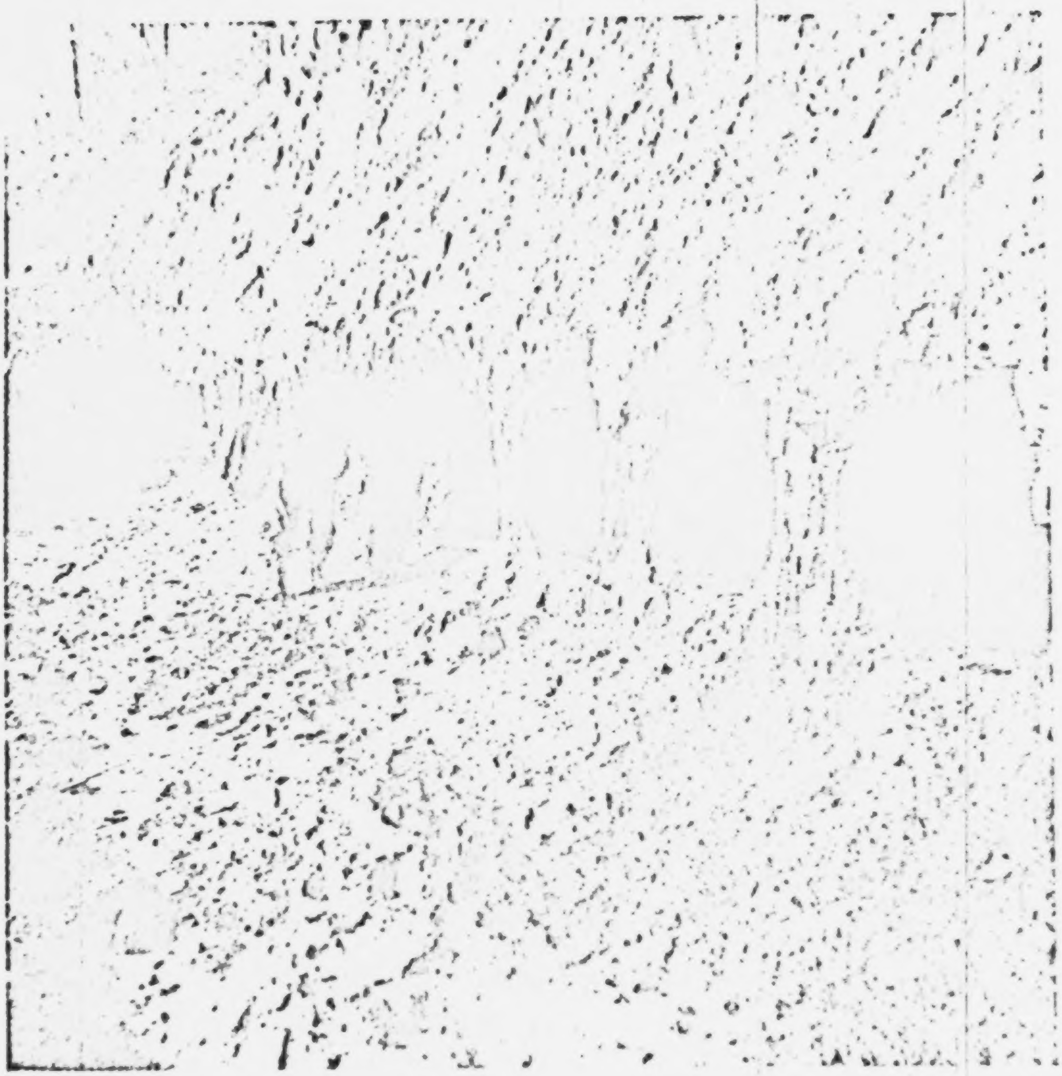


FIGURE 60

Eleven holes were hydraulically cleaned with high pressure water at a pressure of 150 to 200 pounds per square inch. No further holes were cleaned at this time due to a breakage of the hose. Work was then shifted from this mine to another phase of the project and no further work was done here for a period of about 4 weeks.

After the delay, it was necessary to reclean the face of the auger holes due to sloughing and sliding of the high wall. Five of the holes originally cleaned were left as found with about one foot of mud in the bottom of the hole. This mud and debris was left in the front part of the hole, then the area behind this was cleaned out in the place where the plug was to be placed. The remaining holes were cleaned completely, but work was not done until each hole was ready for plugging.

Two types of plugs were used in this remedial work. The first type of plug was a double-wiper wooden plug. The wipers were composed of heavy rubber belting 41 inches in diameter, sandwiched between two plywood discs of 35-inch diameter. The wipers were spaced 4 feet apart with wooden stringers. A hose through one of the discs permitted the interval between discs to be filled with grout slurry. A drain hose extended through the entire plug. A photograph of this plug is shown in Figure 33.

The second type of plug used was a cloth retainer. This retainer had a drain hose extending through its length and a fill hose entering through a small center wooden disc. The cloth portion was then filled with grout slurry to assume the shape of its surroundings. The plug was about 4 feet long when filled.

The plugs were assembled on the site. The cloth retainers required about one hour each to assemble. The wooden type plug required about three hours to assemble.

The first plug was installed in Hole No. 19. The hole was inspected and a good seating area selected. The hole was hydraulically cleaned in preparation for the plug. A wooden plug was then pushed into the hole. The plug fit snugly in the hole and the wipers contacted the wall at all points around the circumference of the hole. However, the weight of the plug caused the plug to settle toward the bottom of the hole and lessened the contact at the upper surface, causing a leakage at the upper level during the pumping of the slurry.

The grout slurry consisted of 50/50 Cement-Fly Ash mixture with 18% Salt, 2% Calcium Chloride, one pound of flake cellophane per sack of Cement-Fly Ash mixture, and 5.05 gallons water per bulk cubic foot of dry material (56 parts Cement, 44 parts Fly Ash, 2 parts Calcium Chloride, 1 part Flake Cellophane and 50 parts water by weight). The slurry was mixed with a Moyno pump and pumped with a reciprocating type pump capable of pumping 175 gallons per minute at 500 psi pressure. The pump was powered with a 140 HP engine. After pumping for 65 minutes, a leak occurred on the upper level and pumping was stopped. The leakage was stopped on the front side of the plug by mechanically closing the points of flow, but leakage continued on the back side. This had been a continuous fill to this point, with the level about 5 inches from the top of the hole, when the leakage occurred. Pumping was stopped at this point. Figure 61 is a photograph which shows this plug in the hole.

Another wooden plug was placed in Hole No. 17 and filled in the same manner as the first wooden plug. The plug was filled using the mixture of 50/50 Cement-Fly Ash with 18% Salt and 4% Calcium Chloride to improve set time. The plug was about three-fourths full in 28 minutes pumping time when it began to leak around the wipers in the upper portion. Pumping was stopped. Subsequently, additional material was pumped into the upper void after plugging the front side mechanically, but slurry leaked out through the back side, so it was not possible to finish filling the upper void.

Hole No. 18 was then selected and hydraulically cleaned. A cloth retainer was placed loosely in the hole in the desired area and pumping was initiated to fill the retainer to the contour of the hole. Slurry was the same as used in the wooden plug in the previous hole. The retainer filled and expanded as expected

FIGURE 61

to about 80% of the volume of the hole. At that time, the one-inch plastic drain pipe parted at the front of the retainer where it was held in place with a hose clamp. This permitted the cloth retainer to elongate, pulling the plastic pipe out of the center wooden disc, causing a leak through this hole. Pumping was stopped while the hole was plugged and then resumed until the retainer was filled within one inch of the top. At this point, the banding around the center wooden disc failed, and the disc blew out of the retainer, causing it to collapse. The retainer was removed and the hole was then cleaned out so it could be plugged again.

Another cloth retainer was set in Hole No. 16 using the same method of placement as on the previous hole. The slurry for this application was also 50/50 Cement-Fly Ash mixture with 4% Calcium Chloride and 18% Salt (56 parts Cement, 44 parts Fly Ash, 4 parts Calcium Chloride, 9 parts Salt and 50 parts water by weight). The retainer was 75% full in 20 minutes pumping time, when it suddenly elongated and fell to about 60% of its height. Then the grout slurry started coming out of the one-inch drain line which was then plugged. This indicated that the drain had either parted inside the retainer or pulled out of the gathered fabric at the back of the retainer. An attempt was made to examine the back side over the top of the retainer, but nothing could be determined, so pumping was discontinued. After this retainer had hardened, another retainer was placed on top of the first and filled to the contour of the hole. This completed operation is shown in Figure 62.

At this time, it was suggested by the Project Officer that the retainers be modified to eliminate the failure of the plastic drain pipe and banding material. The banding around the center wooden disc was replaced with a continuous double wrap to prevent breakage of the banding material. The cloth around the rear portion of the one-inch drain pipe was fastened with two 2-inch hose clamps. A 3/4" x 48" steel bolt was inserted through the plastic drain pipe and secured against the pipe at each end with nut, washer and steel plate against the wooden disc to prevent pipe failure through elongation. A sketch of this modified plug is attached as Figure 63. This retainer was then placed in Hole No. 24 after the area was hydraulically cleaned. Pumping was started and continued until the retainer completely filled the auger hole. The retainer was pumped back against the top of the hole several times because of shrinkage due to loss of water from the grout slurry filtering through the fabric. The modified retainer worked successfully to plug the opening.

The next hole selected and hydraulically cleaned was Hole No. 23. Another cloth retainer was modified differently for this application. The cloth sleeve was shortened by three feet and sewed together at the back end using a quadruple chain stitch seam. The drain pipe was plugged inside the retainer. An extra

FIGURE 62

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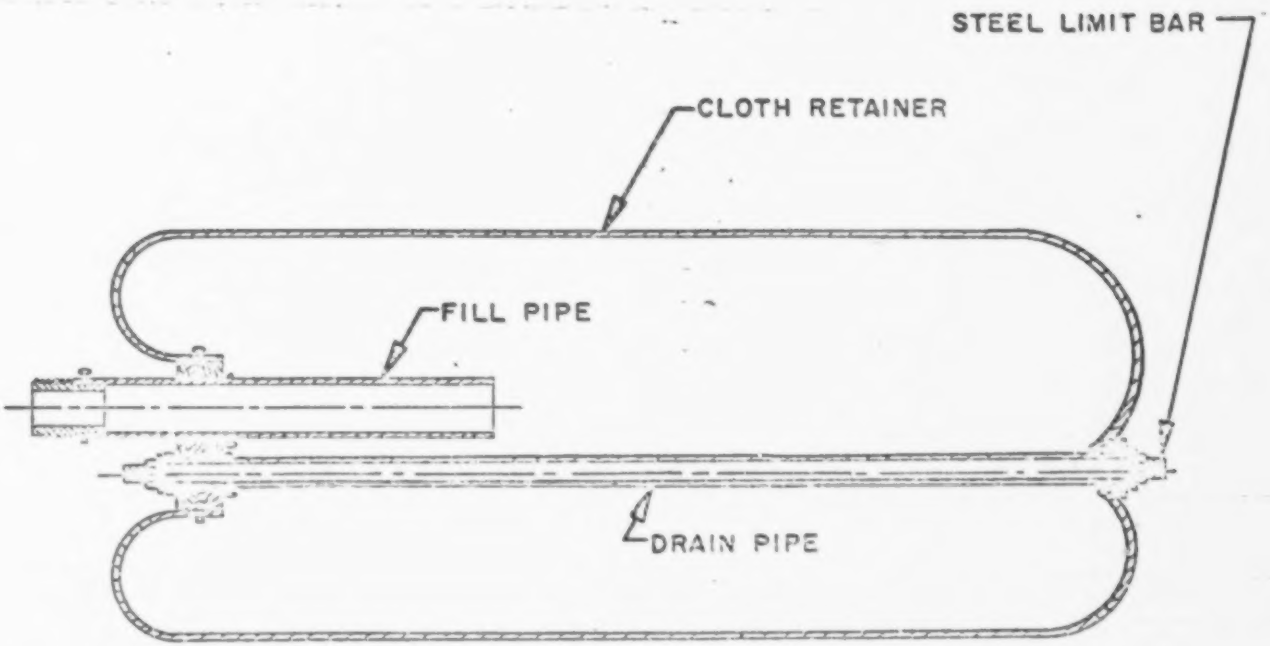

STEEL LIMIT BAR
CLOTH RETAINER
FILL PIPE
DRAIN PIPE

MODIFIED AUGER HOLE RETAINER

FIGURE 63

wrap of banding material was placed around the outside of the cloth on the wooden center disc. Pumping was then initiated. When the retainer was about 80% full, the cloth burst near the wooden disc. This retainer was then removed from the auger hole.

Based on previous test results, a decision was made to continue the hole plugging by using the cloth retainer constructed according to the first field modification, using the steel bar limiting device to prevent elongation.

The next cloth retainer was modified in this manner and placed in Hole No. 18 after the hole was prepared by hydraulic cleaning. This retainer was filled to the top when it apparently leaked on the back side and fell about 6 inches in height. This was then allowed to partially set and then refill. This procedure was repeated two more times until the void was filled completely.

Hole No. 23 was then cleaned and prepared again. A cloth retainer, modified with a limiting device, was placed in the prepared area. The retainer was completely filled in 15 minutes. Subsequent pumping for brief intervals was necessary to keep the retainer filled due to filtration of water through the fabric. On the third and final pumping, a split about 10 inches long occurred in the top front portion of the cloth, but this did not cause a failure because the grout had already dehydrated in that area. The grout hardened to hold the plug in place successfully. This plug is shown in Figure 64.

FIG. 64

The next auger hole to be plugged contained a man-made drain in the bottom approximately 10 to 12 inches wide. This drain was about 12 inches deep at the front of the auger hole and sloped upward into the rear part of the hole. This drain was cleaned and two clay dams placed 5 feet apart, with the front dam about 5 feet into the hole. This was filled with grout slurry to the contour of the hole and allowed to harden before placing the retainer in the hole. From this point, the job proceeded in the same manner as the previous job described. A photograph of the completed plug is shown in Figure 65.

The remaining auger holes were plugged using the cloth retainer modified with the steel limiting device, with the exception of Hole No. 21. This hole intersected Hole No. 20 behind the cloth retainer installed in that hole, so Hole No. 21 was left unplugged so the rear side of the retainer in Hole No. 20 could be examined if desired. In each case the plug was placed about 5 feet from the front of the auger hole. Only one other failure occurred in the cloth similar to those already described. All other plugs were completed satisfactorily. In each case, several final stages of pumping were necessary to fill the top few inches of each retainer due to permeability of the cloth permitting the exfiltration of water. This was not undesirable since this allowed the grout to harden sufficiently to maintain the required seal in a shorter period of time.

FIG. 65

It is believed that the failures of the fabric occurred primarily because the retainer was not large enough in circumference to compensate for wrinkles and folds. The diameter of the cloth sleeve was 42 inches for use in an auger hole of approximately 38 to 41 inches diameter.

It should be pointed out that the increase in efficiency, because of the improvement of technique as the tests progressed, permitted the pumping time to be decreased from over an hour initially to about 15 minutes for the final plugs which were set in the auger holes.

Due to most auger holes having uneven circumference, caused by chunks of coal breaking out, it was necessary to select carefully the holes in which the wooden wiper plugs could be used. The area for the wipers had to be smooth enough to provide a suitable seat in order to insure a seal. This factor would limit the widespread use of this type plug.

It is recommended that future auger hole plugs be constructed like the modified plug (see Figure 63) with the cloth diameter equivalent to 120% of the hole diameter. In this case, this would cause a cost increase of $3.00 over the present cost of approximately $23.00 for the plug, exclusive of placement costs.

E. Water-level Recorder Installation
Mine No. 14-042-A

With the concurrence of the Project Officer, a substitution was made in the proposed field work. Rather than flooding a mine as set forth in the original scope of work, because of prohibitive cost estimates, the installation of a continuous water-level recorder on Mine No. 14-042A was substituted.

The water-level recording device was to be installed in order to provide a continuous evaluation of the effectiveness of the mine plug. The data as recorded would reflect the changes in water level in the mine so that a comparison could be made between the change in water level and the effect of the rainfall, and therefore ground water entrance into the mine. It would also be helpful in providing data on the relation of the leakage through and around the plug to the height of the water trapped in the mine.

It was necessary to obtain an extension of the original access agreement to this mine site to make this installation and provide time for monitoring and maintenance. This extension was obtained prior to proceeding with the installation.

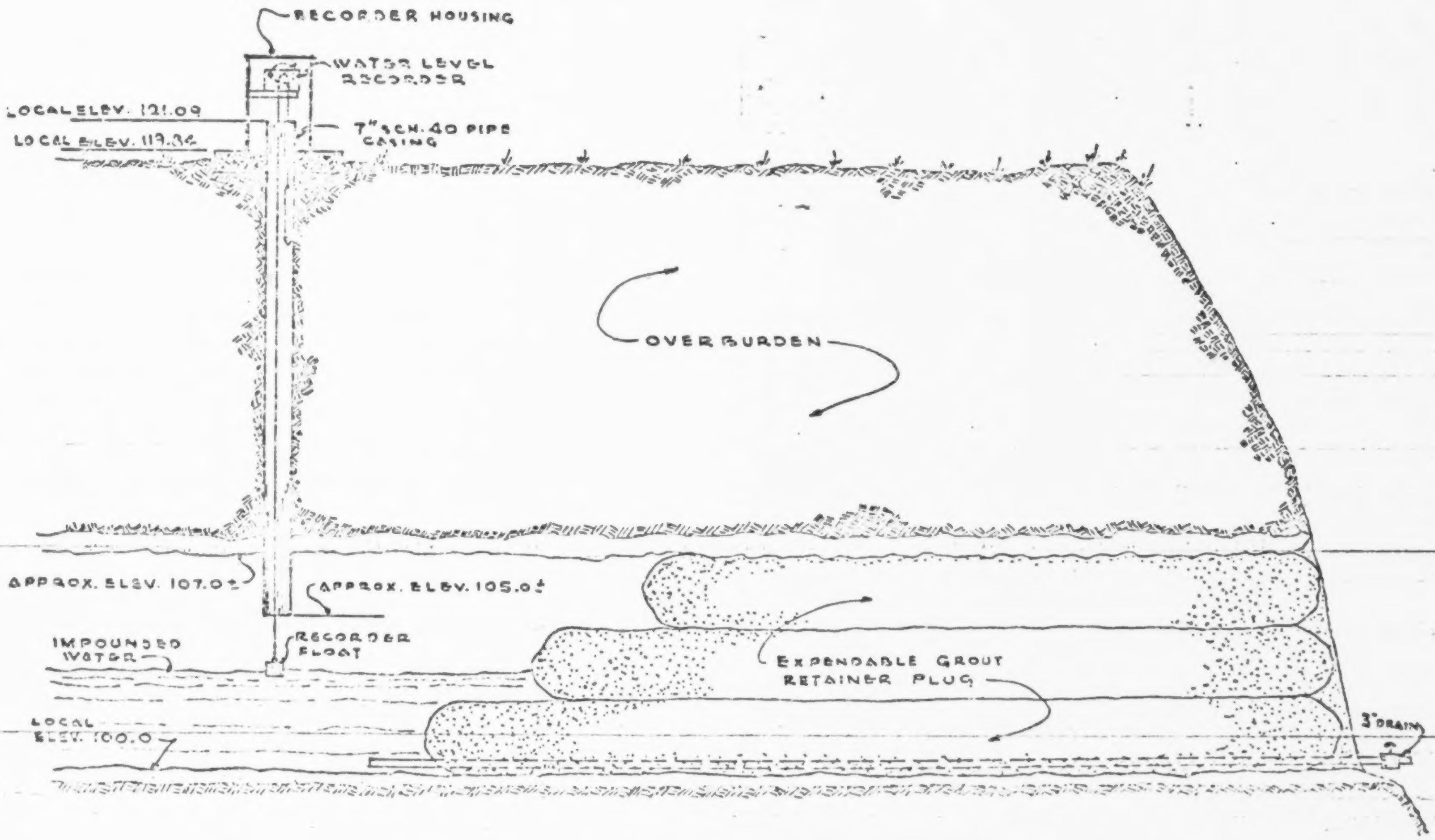
Several types of water-level recording devices were investigated. A Stevens Type F Water-level Recorder was selected and ordered for installation on Mine No. 14-042A. This recorder utilizes a weight-driven clock drive with a 32-day chart. This mine had previously been plugged by the use of Expendable Grout Retainers. A hole had been drilled through the roof of the mine and 7-inch casing cemented in the hole with the lower end extending just below the roof of the mine. The recorder was to be installed on this casing. A drawing showing a section through this mine is attached as Figure 66.

A protective cover was designed, constructed and installed on the casing. Figure 67 shows a view of this installation. A close-up view of the recorder and housing is shown in Figure 68.

After the installation, the protective cover was insulated in order to reduce condensation of warm vapors that were being emitted from the mine through the casing. A desiccant pack was placed in the recorder to aid in keeping moisture off the recording chart.

Two sampling tubes from the drift were plugged with cement, leaving the drain line as the only remaining line which could be opened to flow with a valve.

Water-level data for the mine is shown in Table XIII. A representative sample of the recorder chart is attached as Figure 69. It can be noted how easily the data can be correlated to show the effect of the rainfall and snowfall on the increase of the water in the mine by consolidating the rainfall, snowfall and temperature information with the chart record. It can also be seen that the greatest increase in this particular chart took place immediately following a 3/4-inch rain which took place on January 30. The rise in the water did not level off


RECORDER HOUSING
WATER LEVEL RECORDER
LOCAL ELEV. 121.09
LOCAL ELEV. 119.34
7" SCH. 40 PIPE CASING
OVERBURDEN
APPROX. ELEV. 107.0±
APPROX. ELEV. 105.0±
IMPOUNDED WATER
RECORDER FLOAT
EXPENDABLE GROUT RETAINER PLUG
LOCAL ELEV. 100.0
3" DRAIN

SECTION THROUGH MINE NO. 14-042 A
FIGURE 66

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FIG. 67

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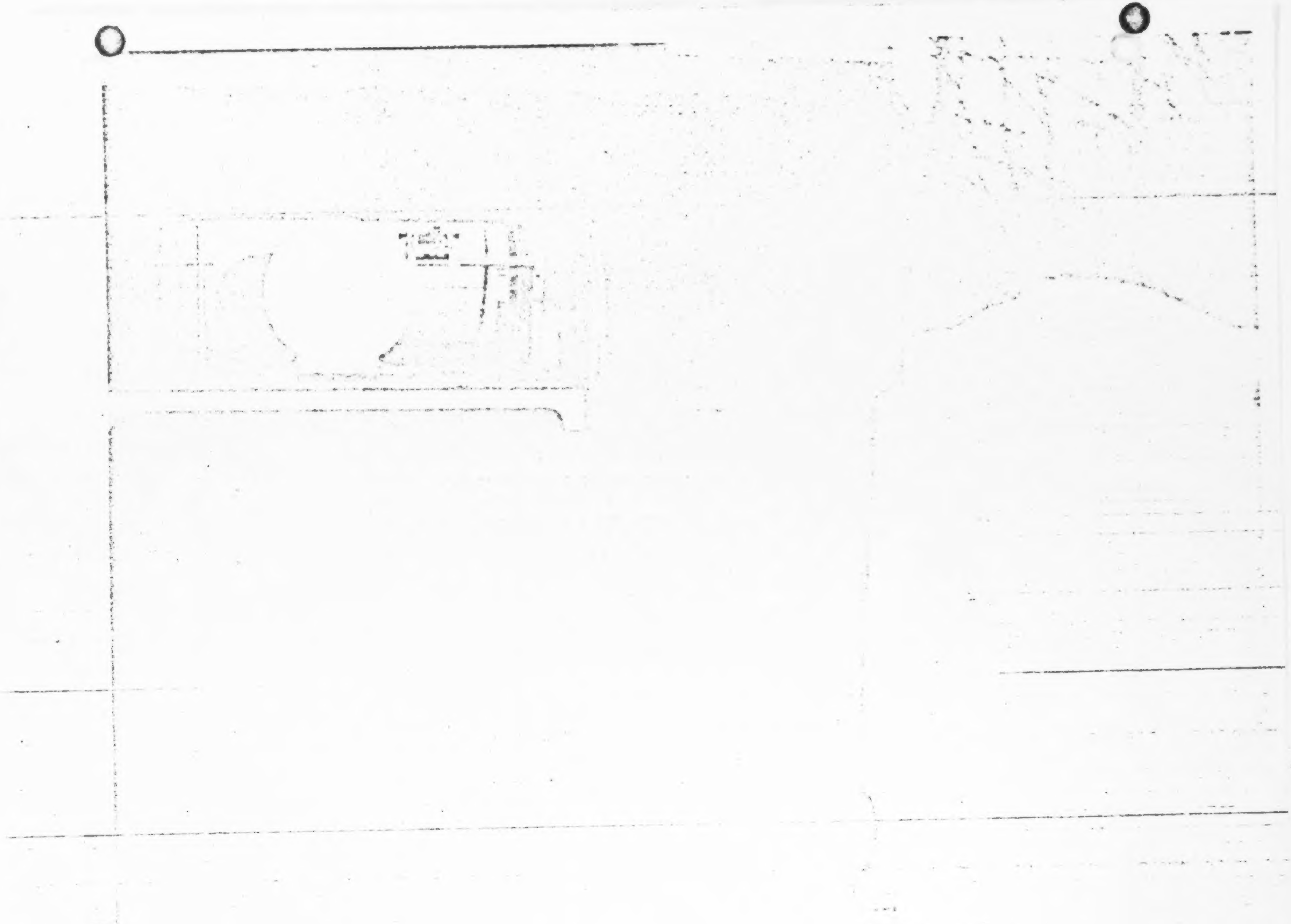
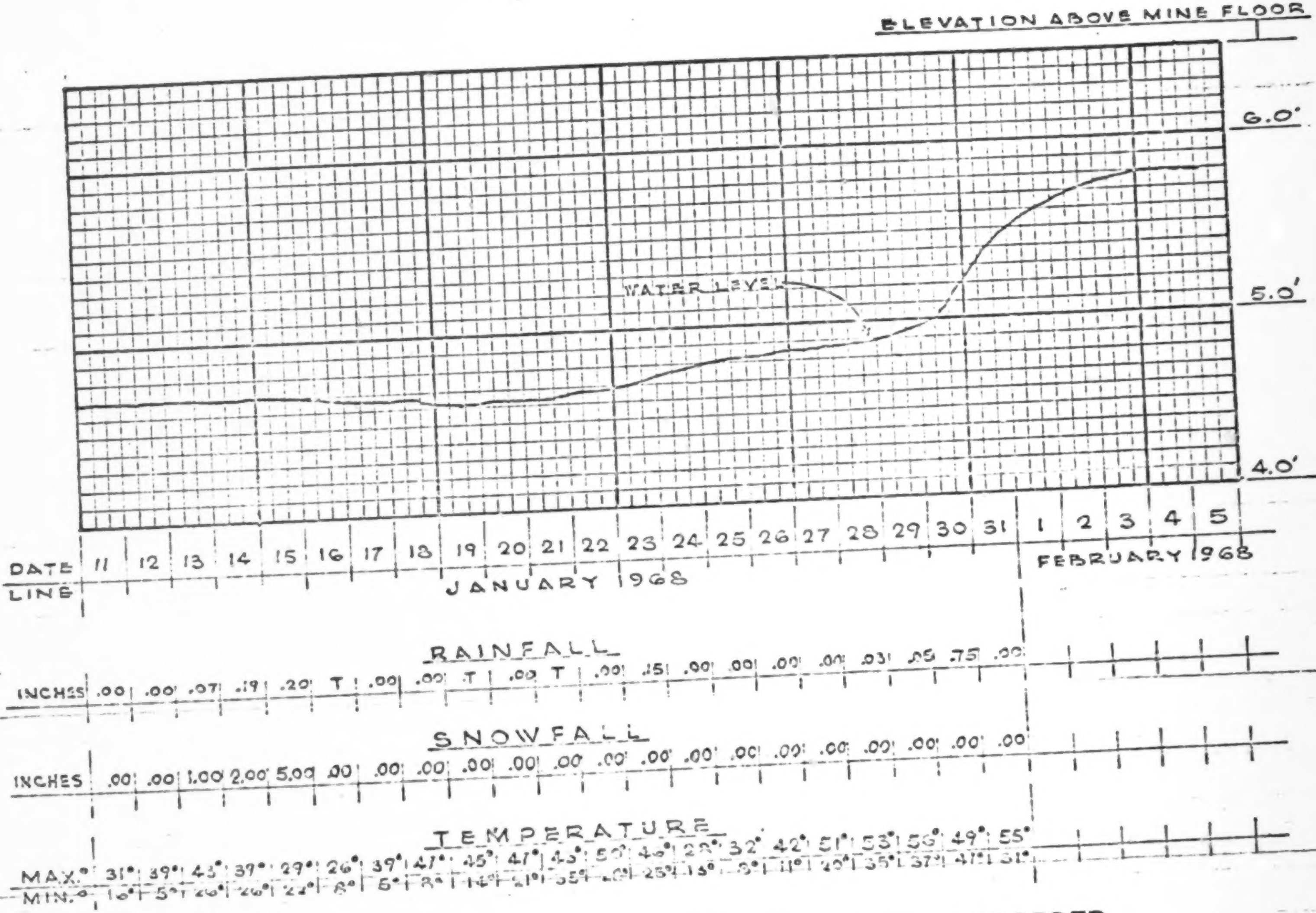


FIG. 68

TABLE XIII

WATER LEVEL DATA
MINE NO. 14-042A

Date	Water in Mine, Feet	Remarks
5-15-67		Closed Valve on Discharge Line
5-20-67	4.93	Manometer Measurement
5-24-67	5.44	Manometer Measurement
5-25-67	5.5	Manometer Measurement
6-6-67	5.71	Manometer Measurement
6-14-67	5.5	Manometer Measurement
6-30-67	5.21	FWPCA Reading
7-17-67	4.83	FWPCA Reading
11-15-67	4.05	FWPCA Reading
11-29-67	4.0	Installed Water-level Recorder
12-7-67	4.0	Balance of Readings from Water-level Recorder
12-31-67	4.8	
1-2-68	4.7	
1-11-68	4.7	
1-27-68	4.83	
1-30-68	5.0	
1-31-68	5.25	
2-1-68	5.5	
2-2-68	5.8	
2-26-68	5.3	



REPRESENTATIVE CHART WATER-LEVEL RECORDER

FIGURE 69

until February 4, indicating the time required for the water to infiltrate into the mine. A check of the mine on February 26, 1968 showed a water level of 5.3 feet in the mine and a leakage of 1.61 gallons per minute measured at the culvert about 30 feet from the mine face. A leak on the left side of the mine plug at a height of 4.83 feet above the base of the coal seam was flowing 0.26 gallons per minute. This flow is apparently coming through the coal seam from the water trapped in the mine. Part of the flow measured at the culvert could have resulted from melting snow, but most of it was apparently coming from the water trapped in the mine.

Since the period that was monitored after the installation of the water-level recorder covered a time in which the weather was below freezing, it was not conducive to drainage of water. We have not obtained the results that will be forthcoming during the spring season when the snow is melting and the rainfall increases. A continuous monitoring during that period will probably give a better correlation than we have obtained through the limited time which we have observed this mine with the water-level recorder installed.

IV. SEISMIC SURVEY AND STUDY

Summary

The objective of the seismic study was to conduct a qualitative study applicable to locating hidden mine openings and thin high wall sections.

The work basically involved location of mine workings from above the mined area utilizing energy in the near vertical plane and location of mine workings from the surrounding high wall utilizing energy in the horizontal plane. This work may be enumerated as follows:

(1) Location of mine workings utilizing high frequency reflection or refraction techniques.

(2) Location of mine workings near the high wall created by strip mining operations and definition of the thickness of the high wall to such mine workings using high frequency reflection seismic techniques.

An unmined or control area was selected on a property in the near vicinity of one which contained mine workings. Seismic data from this control area was compared to seismic data from the mined area. A two-line grid was surveyed on the control area and a fourteen-line grid was surveyed above the mined area in which both the Pittsburgh and Redstone coal seams were extensively worked by deep mining methods. Upon these two areas were performed studies using conventional seismic techniques of both reflection and refraction types. Reflection techniques utilized were primarily aimed at location of individual mined areas; whereas, the refraction techniques were directed toward determination of gross mined areas, areas of subsidence or roof collapses and fracturing extending into formations above the mine workings. A control area which had been strip mined, but was not deep mined, was used to provide background data for comparison to two strip mined areas which had been deep mined. Seismic studies of these areas were made in order that results could be compared to yield information regarding the presence and proximity of mine workings near the coal face exposed by strip mining.

Field surveying and subsequent computer data processing were performed concurrently in order to note any discrepancies in data obtained during the survey. Field instrumentation capable of recording to 2.0 kHz was utilized. Seismic signals received by geophones placed along the surveyed grids were transmitted to this recording equipment, filtered and recorded on magnetic tape by a tape recorder operating at twenty times its normal speed, as well as being presented on a film recording. The magnetic tape was played back in a computer center at one-twentieth of its recorded rate in order that existing digital filter programs might be partially utilized.

Field production surveying time required 138 hours to gather the data required for this study. An additional 187.5 hours was lost by equipment breakdown and repair. Seismic recording is impossible during rainy periods or in high winds and a total of 143.75 hours were lost due to the weather. Conventional seismic studies were first begun on Lines No. 1 and 2 of the control or unmined area. Modifications in filtering and in the type geophone used, in addition to the explosive charges used, were made as work progressed in this area. Line No. 14 above the mined area was run utilizing these modifications. Of the two high wall sites selected for their close proximity to deep mine workings, the first was found unsuitable because of fracturing and condition of the exposed coal face. The second site proved suitable for drilling and placing of the seismic charges used.

To gain formation velocity data which could be used for computer correction of seismic obtained velocities, a well was drilled through pillars existing in both the Redstone and Pittsburgh coal workings. A two-receiver acoustic velocity log utilizing one foot spacing between receivers was run in the uncased liquid-filled borehole. The continuous velocity recording was integrated to obtain travel times from the surface to the various formations which existed beneath the surface. The Birdwell Division of Seismic Service, Inc. was employed to perform these measurements. Reintegration of the velocities obtained was necessary to produce realistic travel times.

The Western Geophysical Company's report concludes that the high frequency seismic techniques utilizing shallow reflections provided the most promising results for the diagnosis of the presence of mine workings. The processed data from refraction shooting provided little information regarding presence of mined areas or location of subsidence areas. The lack of reflections shown after processing the data obtained from the unmined high wall is encouraging; however, the data obtained from the mined high wall area also exhibits no continuous reflections. Further improvements in equipment, field operation and data processing are necessary before utilization of seismic techniques may be applied to locate and satisfactorily define areas from either plane.

Equipment improvements which are necessary for further studies are:

a. Improved high frequency response of the geophones used.

b. Improved filters for removal of predominant low frequencies.

c. Calibrated amplifier gains for the geophones used.

d. Tolerance of amplifiers, recorders and playback equipment to ambient temperatures and humidities encountered under field conditions.

e. Redesign recording equipment to provide for higher portability.

Computer programs capable of removal of low frequencies and improvement of signal to noise ratio should be written. Future work in the Clarksburg area should include additional shot holes drilled to the 12,000 feet per second refractor to provide correction data necessary for improved analysis.

The results of the seismic study were, in general, inconclusive. At this time, using presently available equipment, mine workings cannot be located using seismic techniques.

A copy of the report by Western Geophysical Company of America, Los Angeles, California entitled "A Seismic Study for the Halliburton Company" is attached as Exhibit "B", Appendix, page 182.

V. CONCLUSIONS AND RECOMMENDATIONS

As a result of the tests and evaluation performed under this contract, a number of conclusions were made. Several of these are shown in the applicable sections but will be repeated here along with others which are drawn for the purpose of clarity. It is concluded that:

(1) A fast-setting grout slurry was developed which will permit the placement of an Expendable Grout Retainer plug in a drift opening in a one-day operation. The recommended slurry is composed of 30 parts Cement, 30 parts Fly Ash, 40 parts Gypsum Cement, and 60 parts water by weight, with 0.6% retarder added to control set time. It costs $20.66 per cubic yard, exclusive of charges for blending, transporting, mixing or pumping.

(2) A grout slurry was developed which had expansive properties for use between bulkheads in a mine drift. The slurry recommended contains 56 parts Cement, 44 parts Fly Ash, 9 parts Salt and 50 parts water by weight, and costs $19.21 per cubic yard exclusive of charges for blending, transporting, mixing or pumping.

(3) The cost of placing a plug in a mine drift using Expendable Grout Retainers was decreased by the development of more economical retainers and faster placement techniques. The cost for retainers was reduced from approximately $2.50 per square yard of cloth in a previous application to $1.25 per square yard. The placement time per retainer placed was reduced from 5 hours to 2 hours after filling.

(4) It is impractical to use a porous type fabric material for a retainer in order to pass slurry to the outside. A material with enough permeability to allow escape of water was selected.

(5) The grout slurries selected (see Conclusions 1 and 2) will bond satisfactorily to their surroundings when a plug is installed in a drift.

(6) The full-scale tests confirm that a sustained load equivalent to the weight of grout slurry contained is applied to the floor and walls by a grout retainer immediately, and further load is applied as the grout expands during setting. In these tests, the load increased from 7,000 pounds after filling to 21,750 pounds after 170 hours.

It is necessary to maintain a slight pressure (less than 3 psi) on the upper retainer to effect a seal on the roof.

(7) The use of aggregate in a retainer with subsequent grouting in place does not promise economic advantages over the use of grout slurry alone, primarily because of difficulty in proper placement of the aggregate within the retainer and the necessity for extra equipment involved in placement.

(8) A cloth-type retainer costing under $25.00 was developed for use in auger holes. The retainer is a special cloth sleeve-type container which expands when filled with grout slurry to conform to the shape of the hole.

(9) Laboratory tests in remotely building a bulkhead, both small-scale and full-scale, were encouraging and pointed the way toward further development work in this area.

(10) The feasibility of placing aggregate pneumatically in a desired remote location was demonstrated by moving approximately 600 pounds per minute of both 3/4-inch and 3/8-inch graded aggregate with a pressure of 20 psi.

(11) The use of a horizontal grout curtain to prevent perculation of water over a large area (about 5,000 square feet) is of doubtful benefit.

(12) Disposal of gob by mixing with water through a cellar-jet and pumping into selected disposal areas was not found to be economically feasible.

(13) A cleaning jet was developed which could effectively clean and prepare a drift opening remotely for application of a mine plug. The jet was constructed of 2-inch pipe with a flattened cleaning nozzle and mounted on a 2-wheel dolly so it could be maneuvered from outside the mine. Cleaning was accomplished by pumping water at 150-200 psi pressure through the jet.

(14) The technique of placing an aggregate-type plug of 3/8-inch and 3/4 to 1-1/2-inch limestone pneumatically and subsequently grouting in place was proven successful in field construction tests. The plug portion, composed of 3/4 to 1-1/2-inch aggregate, was grouted in place using a cement grout consisting of 50/50 Pozzolan Cement with 2%

Calcium Chloride, 18% Salt and one pound of cellophane flakes per sack of solids.

(15) The aggregate-type plug is readily adaptable to most drift openings as shown by placement in a drift containing rows of side and center posts.

(16) Auger holes of approximately 38 inches in diameter can be successfully plugged at the rate of 8 to 10 holes per day using specially designed cloth retainers. A retainer incorporates a 42-inch diameter fabric sleeve so designed that it will plug the hole for a length of about 4 feet when filled with cement grout. Slurry used for grouting was composed of 50/50 Pozzolan Cement with 4% Calcium Chloride and 18% Salt.

(17) The seismic study performed showed that the electronic equipment procedures and data analysis methods were inadequate to define the objectives outlined; but the study pointed to modifications necessary to improve the resolution of this technique.

Based on the conclusions drawn from the study, the following recommendations are submitted:

(1) It is recommended that plugging of drift openings of abandoned mines be accomplished by one of the following procedures after drift has been opened to full width:

A. Utilization of the improved Expendable Grout Retainers and the fast-setting slurry selected in this study. Using a bottom grout retainer of 25 feet (filled), this should provide a plug in a drift opening 8 feet wide and 7 feet high at a cost of approximately $2,758.00, exclusive of supervisory personnel salaries. This cost includes the use of the recommended slurry from Conclusion No. 1.

B. Utilization of an aggregate-type plug similar to that shown in Figure 55. The aggregate would be placed pneumatically from pressurized tanks outside the mine. A back bulkhead of 3/4- to 1-1/2-inch aggregate would be built, then covered with a one-foot layer of minus 3/8-inch aggregate and sprayed with grout slurry to insure seal before grouted section of 3/4- to 1-1/2-inch aggregate and grouting pipes are placed. Grouting pipes should be made with a plugged end and side ports each 18 inches through the section, with only the top and bottom row of pipes placed along with the aggregate build-up. After the required amount of aggregate is placed in the grouted section, an outer bulkhead of minus 3/8-inch aggregate is placed for a minimum thickness of one foot and sprayed with grout slurry as a seal. The grout as shown in Conclusion No. 14 should be used and pumped through the one-inch grout pipes until the voids are completely filled in the grouted section. A plug of this type, 30 to 34 feet in total length, should cost approximately $3,200.00, exclusive of supervisory personnel salaries.

(2) It is recommended that auger holes be plugged using the modified fabric-type retainer shown in Figure 63, and the slurry as shown in Conclusion No. 16 which was proven satisfactory in field application. A plug of this type in a 40-inch auger hole will cost approximately $130.00, assuming that 8 to 10 holes will be plugged in an eight-hour shift.

(3) It is recommended that preparatory cleaning of auger holes and mine drifts be accomplished using the maneuverable jet described in Conclusion No. 13.

(4) It is recommended that further development work be undertaken in laboratory and field tests for remote building of bulkheads using a quick-setting type grout material.

(5) The placement of a horizontal grout curtain to prevent perculation of water is not recommended.

(6) Disposal of gob by mixing with water and pumping to a disposal area is not recommended.

(7) Additional seismic study is not recommended using equipment currently available.

Browns Creek
[illegible]
study of 10-062

APPENDIX

WEATHER CONDITIONS
Clarksburg. West Virginia

MONTH OF OCTOBER, 1967

Date	Temperature, °F. Max.	Temperature, °F. Min.	Rain, Inches	Snow, Sleet, Hail Inches	Atm. Cond.
1	71	47	0	0	Clear
2	79	40	0	0	Clear
3	80	46	0	0	Clear
4	79	48	0	0	Clear
5	81	53	T*	0	Cloudy
6	66	44	.12	0	Cloudy
7	61	46	T*	0	Cloudy
8	75	50	T*	0	Cloudy
9	67	51	.05	0	Cloudy
10	62	48	0	0	Cloudy
11	52	44	0	0	Cloudy
12	56	34	0	0	Cloudy
13	68	32	.06	0	Pt. Cloudy
14	69	44	0	0	Cloudy
15	79	42	0	0	Pt. Cloudy
16	76	55	0	0	Cloudy
17	75	49	0	0	Cloudy
18	61	42	.75	0	Cloudy
19	49	34	T*	0	Cloudy
20	59	28	0	0	Clear
21	60	34	0	0	Clear
22	61	26	0	0	Clear
23	73	28	0	0	Clear
24	71	36	0	0	Clear
25	64	41	.72	0	Cloudy
26	51	34	0	0	Pt. Cloudy
27	60	34	0	0	Cloudy
28	46	32	0	0	Cloudy
29	57	27	0	0	Clear
30	72	28	0	0	Clear
31	67	37	0	0	Cloudy
Avg.	66	40			
Total			1.70	0	

* Trace

EXHIBIT "A"

DIGITAL DATA PROCESSING

GENERAL

All of the digital data processing was designed to retain diagnostic data in as undisturbed condition as possible while minimizing the various types of energy which tend to interfere with diagnostic data. Additionally, the processing was designed to simplify the diagnosis by removing non-diagnostic time variations caused by topography, near-surface velocity variations and spread geometry. Finally, the processing was carried through those mechanical steps necessary for the machine-plotting of continuous time-sections in the form of a sequence of variable-area traces.

EQUIPMENT USED

Western Geophysical Transcriber, Model No. D-3, capable of handling SIE tape, either AM or FM, and Western-Techno (AM) tape. Western-Techno tape was used on this job. Normal and half-speed transcription is provided.

One Cathode-Ray oscilloscope Model No. VM-1-S, capable of displaying and retaining 24 traces of seismic signal plus 4 traces of control data for any selected portion of a field record.

Western Geophysical Camera Serial No. 153 providing a permanent "wiggly-line" paper recording with time-break, time signal, time-scale grid, and 24 traces of seismic signal. (Corresponds to an old-fashioned "wiggly-line" seismogram.)

Redcor Multiplexer and A/D Converter Model 600, capable of recording up to 28 channels of 16-bit information in 28-fold multiplex on ½-inch, 7-track, digital tape with 800 bits-per-inch packing.

IBM Digital Computer Model 360, capable of rapidly handling far in excess of the requirements of this job.

Redcor D/A Converter Model 600 capable of 24 channels of simultaneous data output plus 4 channels of simultaneous control information.

Western Geophysical seismic Amplifier Model No. FA 50, capable of amplifying, filtering, and setting gain level on 24 channels of seismic data and 4 channels of control signals.

Section plotter Model No. SP-112-1, capable of plotting up to 240 consecutive traces at wide trace-spacing (as here used) or up to 480 consecutive traces at narrow trace-spacing over an "apparent" time of 6 seconds. (In oil exploration work this is "real" time; in mining exploration and certain engineering studies this "apparent" time converts to a shorter "real" time by the ratio of the (low) transcription tape speed to the higher recording-tape speed.) This plotter can provide variable-area, variable density, "wiggly-line", or combination display modes together with a time grid.

PROCESSES EMPLOYED (in sequence)

1. Transcription from field analog magnetic tape to 28-trace multiplexed 16-bit digital tape.

2. Editing and Adjust-Editing to test digital accuracy and correct for any variations in the speed of the field analog tape.

3. Reformating to de-multiplex and store channel-by-channel on digital tape.

4. Applying "Optimum Amplitude Recovery" to approximately equalize the amplitudes on all channels.

5. Applying Normal Moveout and Static corrections to remove the effects of topography, shallow vdlocity variations, and spread geometry.

6. Performing 600% Common Depth Point stacking to accentuate primary reflections over various undesirable seismic energy. (When 600% coverage was shot.)

7. Section Plotting with one or more filters in the combination variable-area, "wiggly-line" mode, with time grid.

Note: In one instance, Highwall Line 2, the noise level was so high that step 4., "Optimum Amplitude Recovery" was replaced with a broad digital filtering process which included a mild form of OAR, but did not tend to bring the level on weak channels (mostly noise) up to the level on the strong channels (mostly signal).

It should be pointed out that mixing (or summing) of adjacent traces before stacking (or of adjacent stacked common-depth-point traces after stacking) was avoided. If this general approach to locating tunnels or any other very localized anomalies is to be successful at all, such mixing must be avoided, since it would tend to average anomalous information with normal information and "blur" the anomaly. Common-depth-point stacking, however, does not "blur" the anomalies in this fashion, since every component trace in the stack is reflected from the same localized part of the reflector.

RESULTS AND CONCLUSIONS

GENERAL

In general, the data indicates that too much low-frequency noise has been recorded. This low frequency, (below 1.0 KHz), is caused by surface waves that over-ride the filters and hide the much smaller high frequency component. In addition to, the low frequency noise there is considerable high frequency crossfeed generated within the recording system. In spite of this noise and crossfeed, it is possible to see indications of legitimate reflections from the zone of interest on both the highwall and surface records.

Those indications of legitimate information from the zone of interest leads to the general conclusion that, with proper instrumentation and geophones, it is possible to record shallow reflections from both highwall and surface shooting.

HIGHWALL

The data from line 1, unmined highwall shows no strong reflected energy at any of the 4 velocity components computed. The few weak lineups observed are most likely caused by minor discontinuities within the coal section.

Since no mine workings are believed to exist at highwall line 1, absence of strong reflections on this line is a result favorable to the problem of diagnosing the presence of hidden mine workings from recorded reflections. This is especially true in view of the fact that the records were processed with 600% stacking to improve the signal-to-noise ratio. It will be remembered (Appendix 3/4 2) that sufficient reflected energy existed on the highwall line 1 records to permit the calculation of velocities. However, these calculations were made from common-depth-point families and thus no continuity of reflector is implied; sporadic, unrelated reflectors or diffractors could provide the energy utilized in these calculations. The lack of continuous reflectors after stacking indicates that this condition probably exists.

Unfortunately, no continuous reflections are obtained at highwall line 2, even though mine workings are believed to exist there. Several circumstances probably contributed to this result: 1) the signal-to-noise level was very low (no reflections or diffractions could be seen even on the common-depth-point families), 2) the highwall at this location was covered with rubble so that the shots and geophones were mostly in the rubble, not in or on the solid coal face, 3) the 600% stacking tended to "blur" the reflections from a particular side of the central adit with those from the other side of the adit. If it were not believed that these records were so noisy as to be well-nigh unusable, a different type of stacking (limited to each side of the adit separately) would have been advisable.

Inspection of the unfiltered, unstacked records at highwall line 2 indicates that further processing involving "zero moveout rejection" (most of the noise is zero moveout), "moveout search", "variable % stacking," and "optimum moveout accentuation," all limited to a particular side of the adit, might be fairly successful in sharpening reflections now nearly hidden in the noise; possible reflection alignments agreeing roughly with those to be expected from the mine workings may be faintly seen crossing through the noise by such inspection.

LINE 1, STUDY AREA NO. 1 (Surface)

LINE 14, STUDY AREA NO. 2 (Surface)

These lines have established that shallow reflections suitable for the diagnosis of mine-workings can be obtained at high-frequencies. They indicate, however, that more sophisticated static corrections will be required before the preferred procedure for complete diagnosis can be said to have been established. On portions of both of these lines, excellent, continuous reflections may be seen. An

MONTH OF NOVEMBER, 1967

Date	Temperature, °F. Max.	Min.	Rain Inches	Snow,Sleet,Hail Inches	Atm. Cond.
1	68	51	.15	0	Cloudy
2	56	46	.45	0	Cloudy
3	59	40	.05	0	Pt. Cloudy
4	49	33	.17	0	Cloudy
5	34	28	T*	T*	Cloudy
6	34	27	0	T*	Cloudy
7	41	23	0	0	Pt. Cloudy
8	42	23	0	0	Clear
9	58	21	0	0	Clear
10	62	30	0	0	Cloudy
11	71	36	T*	0	Cloudy
12	64	46	T*	0	Cloudy
13	48	34	.05	T*	Cloudy
14	39	33	T*	T*	Cloudy
15	35	18	.02	T*	Cloudy
16	37	18	0	0	Cloudy
17	46	33	.25	0	Cloudy
18	47	40	.02	0	Cloudy
19	45	29	0	0	Cloudy
20	38	22	0	0	Cloudy
21	48	33	.07	0	Cloudy
22	54	44	.61	0	Cloudy
23	45	31	.17	0	Cloudy
24	44	24	.17	0	Cloudy
25	49	30	0	0	Cloudy
26	55	30	0	0	Cloudy
27	44	29	0	0	Cloudy
28	34	23	0	0	Clear
29	40	20	0	0	Cloudy
30	35	28	.85	3.0	Cloudy
Avg.	47	31			
Total			3.05	3.0	

* Trace

EXHIBIT "A"

MONTH OF DECEMBER, 1967

Date	Temperature, °F. Max.	Min.	Rain, Inches	Snow, Sleet, Hail Inches	Atm. Cond.
1	34	20	0	0	Cloudy
2	48	20	.41	0	Cloudy
3	49	34	.49	0	Cloudy
4	43	26	0	0	Clear
5	54	30	0	0	Cloudy
6	57	32	.13	0	Cloudy
7	61	44	.38	T*	Cloudy
8	55	31	0	0	Pt. Cloudy
9	51	28	0	0	Cloudy
10	53	36	.30	0	Cloudy
11	57	44	.04	0	Cloudy
12	55	39	0	0	Cloudy
13	55	29	0	0	Cloudy
14	49	34	.10	0	Cloudy
15	55	34	0	0	Cloudy
16	35	22	0	0	Clear
17	52	19	0	0	Clear
18	68	42	.07	0	Cloudy
19	58	34	0	0	Clear
20	59	29	0	0	Clear
21	72	41	.21	0	Clear
22	68	27	.60	0	Cloudy
23	38	19	.02	T*	Cloudy
24	36	17	0	T*	Clear
25	46	18	.10	0	Cloudy
26	38	20	0	0	Cloudy
27	31	13	0	0	Cloudy
28	33	26	.75	8.0	Cloudy
29	30	20	0	0	Cloudy
30	29	18	0	0	Cloudy
31	34	22	.30	3.0	Cloudy
Avg.	48	28			
Total			3.90	11.0	

* Trace

EXHIBIT "A"

MONTH OF JANUARY, 1968

Date	Temperature, °F. Max.	Min.	Rain, Inches	Snow, Sleet, Hail Inches	Atm. Cond.
1	30	-4	0	0	Cloudy
2	29	-7	.02	T*	Cloudy
3	40	30	.22	T*	Cloudy
4	34	14	.05	T*	Cloudy
5	20	-3	0	0	Clear
6	32	10	.19	3.0	Cloudy
7	31	3	.02	T*	Cloudy
8	13	-10	0	0	Clear
9	31	11	.09	.5	Cloudy
10	30	23	0	0	Cloudy
11	31	16	0	0	Clear
12	39	5	0	0	Cloudy
13	43	26	.07	1	Cloudy
14	39	26	.19	2	Cloudy
15	29	22	.20	5	Cloudy
16	26	8	T*	0	Clear
17	39	5	0	0	Clear
18	47	8	0	0	Clear
19	45	14	T*	0	Clear
20	47	21	0	0	Clear
21	43	35	T*	0	Cloudy
22	50	28	0	0	Pt. Cloudy
23	46	25	.15	0	Cloudy
24	28	13	0	0	Pt. Cloudy
25	32	8	0	0	Clear
26	42	11	0	0	Clear
27	51	20	0	0	Pt. Cloudy
28	53	35	.03	0	Cloudy
29	56	37	.05	0	Cloudy
30	49	41	.75	0	Cloudy
31	55	31	0	0	Clear
Avg.	38	16			
Total			2.03	11.5	

* Trace

EXHIBIT "A"

# A SEISMIC STUDY
for
THE HALLIBURTON COMPANY

LOST CREEK, WEST VIRGINIA
11-6-67 to 2-19-68

This Work Authorized Under
FWPCA Contract 14-12-31

WESTERN GEOPHYSICAL COMPANY OF AMERICA
A SUBSIDIARY OF LITTON INDUSTRIES

933 North La Brea Avenue -- Los Angeles 38, California

EXHIBIT "B"

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INTRODUCTION

From November 6, 1967 to January 16, 1968, Western Geophysical Company of America's Field Party M-2, under contract to the Halliburton Company, conducted an experimental seismograph survey of lands located in Harrison County, West Virginia near the town of Lost Creek, West Virginia. Party headquarters were located in Clarksburg, West Virginia during the course of the survey.

The purpose of the survey was to evaluate the seismograph process as a means of locating abandoned, near-surface mine workings in both the horizontal and vertical plane.

All field data were returned to the Western Geophysical Company of America offices in Los Angeles, California for processing and interpretation.

The work reported herein was authorized under FWPCA Contract 14-12-31.

DISCUSSION OF RESULTS

Results of both the surface shooting and the highwall shooting indicate that the instruments and geophones were not adequate. In spite of a 1.8 KHz band-pass filter and a 1.0 KHz low-cut filter, frequencies below 1.0 KHz were recorded. Also, high frequency crossfeed was generated within the recording system. The combination of the two types of noise created a very unfavorable signal-to-noise ratio. By digitally processing the data, the signal-to-noise ratio was improved to a certain extent. It is obvious, however, that the instrumentation must be improved before the seismic method proves to be a useful tool in locating shallow, unmapped, mine workings.

Interpretive techniques have been developed for the reduction of this data. Digital programs have been modified and several new programs have been written where needed. With proper recording techniques, reduction of shallow reflection data would be, for the most part, routine.

Only key portions of the original program were recorded.

Any additional shooting with the recording system used would have been a wasted effort. It was considered wiser to repeat, with such changes to the instruments as could be made in the field, the few lines forming the basis of this report. In this way, it was possible to obtain a maximum amount of useful data.

FIELD OPERATIONS

LOCATION

Two areas were laid out for testing. The first, designated "seismic study area number one", is in an area where no known mining has taken place with the exception of shallow auger mining of the highwall.

"Seismic study area number two" is located over extensive mining of both the Redstone and Pittsburgh coal seams. This area is approximately ½ mile east of study area number one and is on the same hill.

Seismic study area number one consists of two northeast-southwest trending lines 1200 feet long and 50 feet apart. Seismic study area number two is made up of a grid of 14 lines, seven in a northeast-southwest direction and seven in a northwest-southeast direction. All lines are 1200 feet long and 200 feet apart.

EQUIPMENT AND PERSONNEL

The following equipment was used:

1- Western Geophysical Company of America T.F.A. seismic recording system, modified to operate within a frequency range of 100 C.P.S. to 2,000 C.P.S., with associated equipment.

2- Ford Bronco 4 wheel drive trucks furnished by the Halliburton Company

1- Party manager's car, rented locally.

The field staff included:

1- Party manager who directed all field operations.

1- Instrument technician - observer.

3- Helpers furnished by the Pettito Brothers.

WEATHER

Fourteen days were lost because of inclement weather. High winds, rain, and snow were the major causes of lost time. Wind velocities as high as 61 knots and temperatures as low as 15 degrees below zero were recorded at the Clarksburg, West Virginia airport.

Mud and snow slowed down operations considerably, making it difficult to drive on the steep roads or walk along the lines of survey.

Adverse weather conditions contributed materially to instrument breakdown. Moisture caused shorting throughout the recording system, resulting in crossfeed and failure of some circuits.

SURVEYING

Horner Brothers Engineers of Clarksburg, West Virginia were contracted for surveying services. This firm located all shotpoint and geophone positions and established elevations at each shotpoint and geophone. A base map of the survey area was prepared showing the locations of the lines of survey and each shotpoint.

DRILLING

No drills were used in seismic study areas one and two. It was felt that, since the zone of interest is less than 200 feet below the surface, light explosive charges would be adequate and that deep shot holes would not be necessary. Therefore, shallow 30 inch holes were worked into the ground using a heavy wrecking bar. This method was effective so long as the light charges were used.

The Byron Construction Company auger drilled all geophone and shotholes for the highwall phase of the program. This work progressed slowly because of weather and the fractured condition of the highwall. Mud and snow made it very difficult to get the equipment close to the highwall and cold caused the air line from the compressor to the drill to freeze.

SHOOTING

Standard seismic shooting procedures were used throughout the survey. Explosives used were seismic water-work boosters, 60% seismic gel (dynamite), and seismic electric caps with 20-foot leads. The shots were fired by the high-speed magnetic-tape drum using a Western Geophysical Company model 2 blaster. Shot sizes ranged between 1 water-work boosters and 2 pounds of 60% dynamite.

INSTRUMENTATION

A specially modified Western Geophysical Company TFA-1 recording system was used on this project. This recording system included:

2- T.F.A. Boxes, each containing 12 TFA-1 amplifiers, 12 1.8-KHz plug-in filters, 12 T.F.L. filters, and 12 M.C. filters.

2- Power-control boxes.

5- Assorted master control, relay and inverter boxes.

1- 1.0KHz low-cut filter box.

1- S.I.E. portable camera type PRO-11-10.

1- Western Geophysical Company Model PF-28 standard-speed magnetic-tape transport.

1- Western Geophysical Company Model PF-28 magnetic-tape transport modified to operate at 5, 10, and 20 times normal speed.

1- Western Geophysical Company Model 2 blaster.

1- Patch panel to control common-depth-point shooting.

6- 264-foot recording cables with six geophone-takeouts on each cable.

2- 200-foot jumper cables.

24- 3-element F.B.X. 27-cycle geophone units.

1- Photo developing unit.

Assorted test equipment, batteries, and supplies.

SHOT AND SPREAD GEOMETRY

The recording techniques used on this project may be described as normal and special. The normal techniques apply to study areas one and two; the special techniques to the highwall shooting.

Two distinct types of reflection shooting were used in study areas one and two. The first of these resulted in 100% subsurface coverage. Twelve geophone groups were placed 10 feet apart on either side of the shotpoint and connected to the recording cables. The end geophone group to the north or east was designated geophone number one. Each geophone group was then numbered consecutively through 24 with the shotpoint halfway between geophone groups 12 and 13. The shotpoints were 120 feet apart, therefore the distance from the geophone group number 1 to geophone group number 24 was 230 feet (23 ten-foot intervals) and the distance from the shotpoint to either geophone group 12 or 13 was five feet.

Two 200-foot jumper cables were used to connect the recording cables to the instruments. This made it possible to locate the recording truck either on a road or at some other easily accessible and level spot. The jumper cables were plugged into the patch panel, cables from the patch panel plugged into the 1.0-KHz low-cut filters and thence into the T.F.A. boxes.

A line of survey starts with a shotpoint. Whenever possible, shotpoint number one is established to the north or east and the shooting progresses to the south or west. To begin a line, 12 geophone groups are placed at even intervals (10 feet in this instance) between shotpoints one and two. A small charge of explosives is then shot at shotpoint one and the seismic energy picked up by the 12 groups of geophones is recorded. Twelve geophone groups are then placed between shotpoints two and three and, without disturbing the geophones between shotpoints one and two, an explosive charge is fired at shotpoint two. This energy is picked up by 24 geophones with the shot in the middle and the 12 geophones between shotpoints one and two have recorded energy from both shotpoints one and two. Thus, by moving the back half of the spread forward after each shot and shooting the next shotpoint ahead, 100% of the sub-surface is covered.

The second type of reflection shooting results in more than 100% sub-surface coverage. The particular configuration used on this project resulted in 600% sub-coverage with two reflection points between each geophone.

The spread geometry was the same as in 100% shooting, except that shotpoints were established between each second geophone group, and only two geophone groups were moved ahead after each shot. At the beginning of a line, shotpoint 1A would be between geophone groups 2 and 3, 1B between groups 4 and 5, 1C between groups 6 and 7, etc. When shotpoint two is shot, the energy is recorded by 12 geophone groups in each direction. This configuration is maintained to the next to the last shotpoint by moving two geophone groups forward after each shot.

In practice, a full spread is laid out beyond the spread being shot and the geophone pairs are moved ahead at the patch panel mentioned above. Each individual recording cable has six geophone takeouts. Two cables are plugged together to make a twelve takeout cable. By using the patch panel to move geophone pairs ahead, it is possible to physically move six geophone pairs with the associated cable thereby speeding up the shooting and making more efficient use of manpower.

Another spread configuration was used in study area number one. All 24 geophone groups were kept ahead of the shotpoint instead of having 12 groups on either side of the shot point. No advantage was gained with this configuration so it was abandoned.

Refraction profiles were shot in both study areas. The spread was positioned with 12 geophone groups on either side of a shotpoint. Instead of detonating an explosive charge at the shotpoint at the center of the spread, charges were fired at the shotpoints at either end of the spread. With the spread in the same position, shots were fired at the next shotpoint away from the ends of the spread and then at two shotpoints away from the ends of the spread. As an example, consider that the spread is positioned between shotpoints 3 and 5 with shotpoint 4 in the middle of the spread. Refraction shots would then be fired at shotpoints 3 and 5, then at shotpoints 2 and 6, and finally at shotpoints 1 and 7. This would complete the refraction shooting for this spread. The 12 geophone groups between shotpoints 3 and 4 would then be moved forward between shotpoints 5 and 6. Shotpoint 5 would now be in the middle of the spread and the shooting procedure repeated. In practice, the refraction shooting is done concurrently with the reflection shooting.

In shooting the highwall portion of the project, it was necessary to project the spread geometry into the verticalplane. Three highwall areas were selected for shooting. Two were in areas of extensive mining and one in an unmined area. One of the mined areas was abandoned because the coal was so fractured, it was impossible to drill shot and geophone holes.

Let us take the final mined highwall shooting as an example. Twenty-four holes were drilled in the coal face 8 feet apart. A fan adit was located between geophone positions 12 and 13. Ten shot holes were drilled halfway between geophone positions 7 and 8, 8 and 9, 9 and 10, 10 and 11, 11 and 12, 13 and 14, 14 and 15, 15 and 16, 16 and 17, and between 17 and 18. Each shot position was loaded and fired in turn and the signal recorded by all 24 geophones.

RECORDING TECHNIQUES

There are 24 TFA-1 amplifiers in the T.F.A. recording system, one for each geophone group, and 24 1.8-KHz bandpass filters. The T.F.A. system is connected to the high-speed tape transport and to the seismic camera. The seismic camera consists of 28 mirror-galvanometers, 24 of which are actuated by the seismic signal, one recording the shot instant, or time-break, one a 2-KHz time reference, and two not used. The high-speed magnetic tape transport was set to operate at 20 times normal speed so that frequencies between 1.0KHz and 2.0KHz might be recorded and the time scale stretched.

An explosive charge, loaded in a 30-inch hole at the shotpoint, was fired by action of a microswitch on the high-speed tape drum. The seismic signal, after traveling to a reflecting interface and back to the surface, vibrates the coil in the geophone, creating a very small voltage. This voltage is transmitted through the recording cable and patch panel and into the 1.0KHz low-cut filters. All frequencies below 1.0KHz are attenuated at this point. The signal passes into the T.F.A.-1 amplifiers for amplification and into the 1.8-KHz band-pass filters for final filtering. The signal is then recorded magnetically on high-speed tape and optically on photographic paper.

The optically recorded, or "monitor", record assures the observer that the time-break circuit is operating properly.

The monitor record also has sharp "first breaks", the first refracted energy that reaches the geophones. This information is later used in computing weathering corrections.

The magnetic tape that is recorded at 20 times normal speed may later be played back at normal speed. The normal-speed tape transport that is part of the field equipment gives this capability to the observer so that he can be certain that he has obtained a satisfactory recording.

Since the objective of this project was approached from the viewpoint of time anomalies rather than amplitude anomalies, no gain settings or amplifier gain calibrations are submitted. In practice it was found that, since the charge size was limited by the fact that no shot-holes were drilled, it was necessary to record shots at the highest gain available in the amplifiers. When using the 1.0-KHz low-cut filters the magnetic-tape modulation did not exceed 10% of the linear range; when these low-cut filters were not used (line 1, seismic study area number one), this modulation occasionally reached or exceeded 100%.

INTERPRETIVE TECHNIQUES

For Control of Field Shooting

During the period of the field-shooting, monitor records and play-backs made in the field were evaluated in the Los Angeles office of Western to determine the adequacy of the field procedures and instrumentation. Such interpretation comprised visual determination of the spectral amplitude distribution of the recorded signals, as well as the signal-to-noise ratio. This current evaluation was, accordingly, only qualitative but it served to provide a gradual modification of the procedures and instruments to obtain better data as the shooting progressed.

For example, the frequency response was improved by broadening the 1.8 KHz filter to increase the response between 1 and 2 KHz and to remove excess transient-stretching at 1.8 KHz. Though no filtering before the amplifiers was found to be desirable when recording refraction records, a beneficial change for recording reflection records was made by inserting 1 KHz low-cut filters ahead of the amplifiers.

In another "field-control" study, a preliminary interpretation of one of the first highwall recordings made with vertical geophones indicated that reflections and refractions typical of near-vertical travel paths dominated the record. As a result of this study, it was verified by consultation with one of Western's engineers (who earlier had taken part in the design and development of the geophones) that the geophones would function satisfactorily lying on their sides, and subsequent highwall tests were made with the geophones planted on their sides, with the normally-bottom-side facing the highwall.

Another illustration of the value of this current control of field procedure is the detection and cure of an instrumental defect (probably arising from low-temperature, high humidity weather conditions) which temporarily greatly increased the noise level on the records. Related to the same general problem of signal-to-noise ratio is the decision, based on such current control, to increase the charge size substantially over that initially employed.

Finally, as a result of preliminary interpretation of the refraction data initially obtained in the field, it was decided that the effective length of refraction profile should be increased in an effort to pick up a deeper refractor. This ultimately permitted a more accurate reduction to a datum plane below which normal-moveout correction could be applied consistently.

For Control of Data Processing

Initial instructions for digital data-processing required preliminary interpretive evaluation of representative analog playbacks of field tapes. For this purpose the more versatile analog playback equipment of Western's Los Angeles Digital Center was employed to display playbacks on a retentive 28-trace cathode-ray screen and to make sample galvanometer tapes for more prolonged study when deemed desirable.

"Statics" (weathering corrections) and "NMO" (normal moveout) corrections had to be calculated by hand, either from the refraction records or from X^2-T^2 velocity calculations from common-depth-point reflection records, because programs available for digital computation of these corrections are, as yet, only marginally effective in the low-frequency range employed in oil exploration; quite inadequate in the high-frequency range necessarily employed in mining and engineering applications of the seismic method.

Filter choices for final displays required experienced interpretive evaluation of preliminary samples in a broad range of filter settings.

In addition, preliminary play-outs following each step of digital process were carefully studied to insure that each step had been properly performed. This was deemed necessary because some of the computer programs had been developed especially for this project; because the time scale and the frequency range were strange to the computer operators; and because the signal-to-noise ratio in such high frequencies was much lower than the operators were accustomed to handle.

FOR EVALUATION OF FINAL RESULTS

When modern seismic digital data processing is employed, interpretation is confined to a geological-geophysical evaluation of processed, continuous displays of time sections. These contain, in a single display, all of the information, which in earlier days were divided among many separate "wiggly-line" seismograms. At a glance, modern continuous time-sections provide a pseudo-geological cross-section which can be evaluated in the light of known geological conditions. Under ideal conditions, a direct relationship can be found between seismic reflections and geological horizons.

Under normal to adverse conditions, "apparent reflections" (fairly continuous line-ups of seismic events on the time-section caused by reflected refractions, diffractions, or multiple reflections) may also appear. These may be barely visible, or, under adverse conditions, as large or larger than the bona-fide reflections.

Before the development of the continuous-display time-section, the presence of non-reflection energy in conjunction with reflection energy on each individual seismogram required a rare, highly-skilled seismic interpreter to detect. On continuous time-sections, the co-mingling of reflection and non-reflection seismic energy becomes immediately apparent because each type of seismic wave has its own peculiar type of continuity, or lack of continuity, along the entire time-section.

Using modern digital data processing, certain programs, notably normal moveout correction and common-depth-point stacking, are employed to accentuate bonafide (primary) reflections having a particular type of continuity relative to multiple reflections and reflected refractions, which possess different types of continuity.

Though no process has yet been devised to diminish diffracted energy relative to reflected energy, the continuity pattern of diffractions on the continuous time-sections is distinctly different from the continuity pattern of bonafide reflections and can easily be recognized and disregarded. In many seismic studies, moreover, the diffraction patterns provide very useful information regarding the presence and location of faulting.

Random noise is also more easily identified and discounted when seen on a continuous time-section than on a series of individual seismograms. Random noise has no definite continuity pattern as may easily be seen when it appears on a continuous time-section; on individual seismograms, there are too few traces to permit unambiguous determination of continuity under adverse noise conditions. Parenthetically, the common-depth-point stacking process, used to de-emphasize multiple reflections and reflected refractions, also reduces the random noise with respect to bonafide reflections by a factor of the square-root of n, where n is the number of common-depth-point recordings stacked.

experimental, "mixed" version of line 1, Study Area No. 1 has been included simply to indicate how the reflections might extend continuously following a more sophisticated static correction and a common-depth-point stacking to avoid "mixing". For proper diagnosis of such small details as mine workings, mixing must, of course, be avoided or be severly limited.

These results have not shown the nature of the diagnosis required to determine the presence or absence of mine workings: they have simply indicated that with the addition of more sophisticated static corrections, data suitable for such diagnosis can be obtained. The static-correction procedure now envisaged as adequate would require essentially the same reflection and refraction shooting procedure as was employed, with the addition of shots in holes drilled to the 12,000- 1/sec. refractor (in this particular area) at spacings of about a thousand feet. The processing would be extended to include "automatic static correction" based on a reflector below the reference refractor and above the first mined coal. This accurate static determination would follow removal of the gross statics as computed from the refractions and the shooting in the few shot-holes drilled to the reference refractor.

It is expected that when properly-corrected data is obtained, the diagnosis of the presence of mine workings will be by means of amplitude, time, and phase changes in reflectors below that upon which the automatic static correction was based. This is theoretically predictable if reflections at high frequency are obtainable and if static corrections of the required accuracy can be made. This work has shown that high-frequency reflections can be obtained and has determined what must be done to attain the requisite accuracy in static corrections.

RECOMMENDATIONS

INSTRUMENTATION

Seismic signal amplitudes at higher frequencies (1.0 KHz to 2.0 KHz) are small because of the natural filtering of the earth. This signal is easily lost in the background noise generated by the instruments themselves. It is therefore recommended that recording instruments be designed and built with shielding and selected components to minimize high-frequency cross-feed and give as favorable a signal-to-noise ratio as possible. These instruments should also include adequate filtering to eliminate unwanted frequencies from the earth signal.

The geophones that are available were designed for oil exploration and operate efficiently at low seismic frequencies (20 to 80 cycles per second). They have a low response to high-frequency signals. It is recommended that geophones be developed that will adequately respond to high seismic frequencies. There may be accelermeters on the market that may be easily adapted to this work.

One serious problem in reducing the field data has been the small variations in speed in the high-speed tape transport. These speed variations have made it necessary to develop a digital program to remove these variations during the digital process. A tape transport that is designed especially for high speed operation must be included in the recording system.

SHOOTING TECHNIQUES

In general, the shooting techniques used on this project were satisfactory. The only major change, except those that may be made necessary by the new equipment, would be to drill shallow holes for the explosive charges. These holes may best be drilled with a hand-held power auger.

FIELD PROCEDURES

In order to optimize the pickup of the small, high frequency signal, attention should be given to geophone plant and geophone array.

Geophones must be coupled to the ground as firmly as possible. Roots, leaves, loose soil, and other surface characteristics of the same nature, increase the filtering effects of the ground substantially. Geophones are rather delicate, therefore, an assortment of spikes that may be driven into the ground, with a top to which the geophone may be firmly attached, should be provided.

It is possible, by arranging the geophone groups in special geometric patterns, to minimize the pickup of surface waves and other undesirable signal components. These patterns should be worked out for the new geophones.

INTERPRETIVE TECHNIQUES

With the new instrumentation, designed specifically for high-frequency recording and amplitude calibrated, it should be possible to work with signal amplitude and wave shape as well as with times, velocities, and reflections, in order to discover amplitude anomalies that indicate the presence of mine workings.

GENERAL

This project has served a vital purpose in determining more precisely the nature of the operational, instrumental, and interpretive problems involved in evaluating the characteristics of the shallow section in this geological and topographic environment. By taking full advantage of this basic knowledge and inaugurating the changes recommended, the problem of detecting and evaluating anomalies caused by mine working can now be attacked with greater confidence of success.

The conclusions here reached were possible after completing (and in some instances repeating, with modified techniques) only key portions of the planned program. Any additional shooting without making the recommended changes would have been wasted effort; it was considered wiser to repeat, with such changes as could be made, the few lines now forming this report.

Respectfully submitted by,
Western Geophysical Company of America

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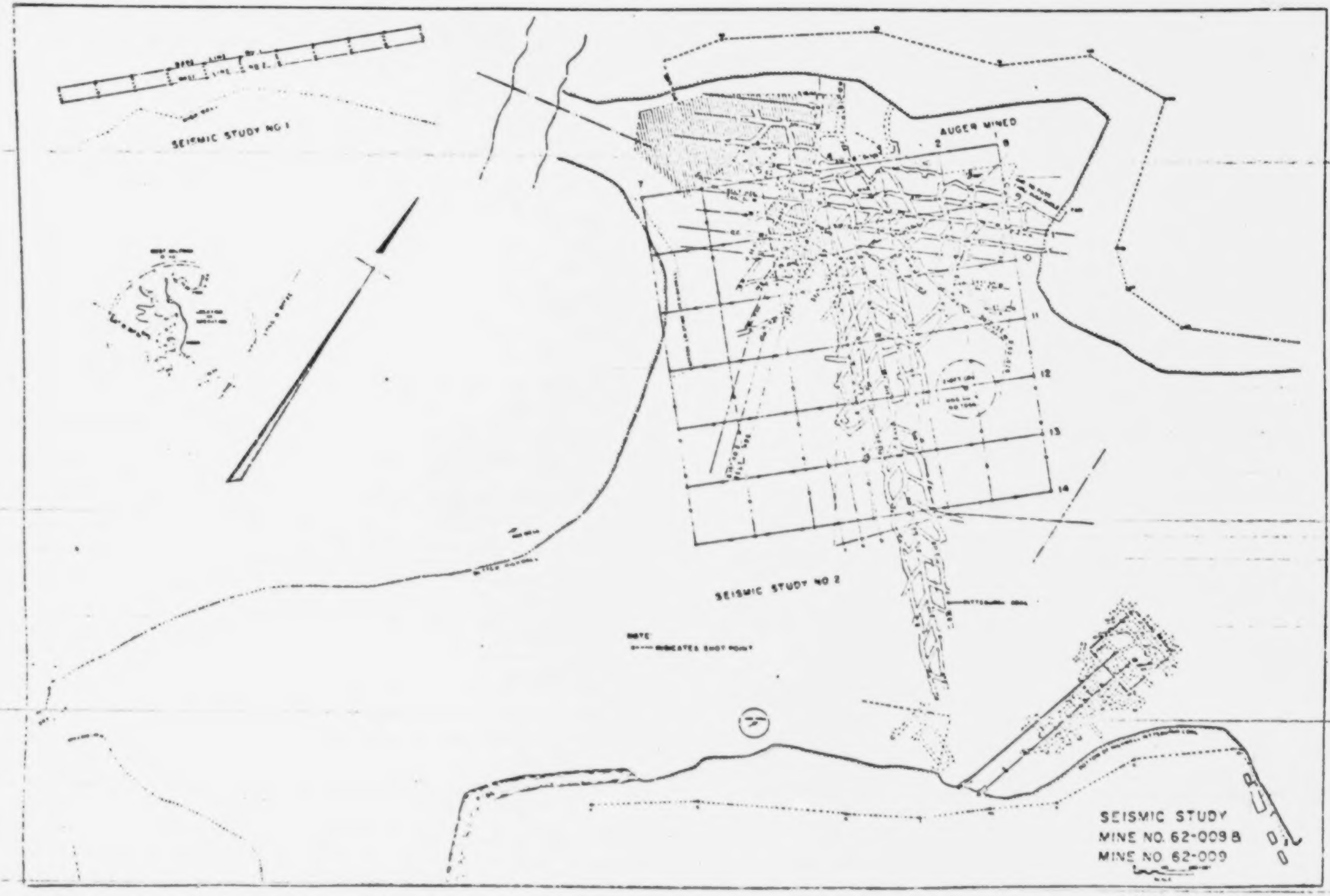
APPENDICES

APPENDIX NO. 1.	Map of survey area showing seismic study areas one and two.
APPENDIX NO. 2.	Summary of Field Operations
APPENDIX NO. 3.	X^2T^2 computations on line 1, unmined highwall. Velocity calculations from common-depth-point families; continuity of a reflector from one family to another was not required.
APPENDIX NO. 4.	Charts show distribution of squared time differences (TX^2-TO^2) in various time ranges. Peaks of the distribution curves used to compute velocity.
APPENDIX NO. 5.	Normal moveout curve for the 2800 foor-per-second velocity, line 1 unmined highwall.
APPENDIX NO. 6.	In separate envelopes. Digital variable-area, "wiggly-line" sections as follows: 600% stack, line 1, unmined highwall; 600% stack, line 2, mined highwall; 600% stack, line 1, seismic study area no. 1; 600% stack, line 14, seismic study area no. 2.


SEISMIC STUDY NO 1
AUGER MINED
2
8
7
11
12
13
14
SEISMIC STUDY NO 2
NOTE
INDICATES SHOT POINT
SEISMIC STUDY
MINE NO. 62-009 B
MINE NO. 62-009

Appendix 2

SUMMARY OF FIELD OPERATIONS

DATE	HIGH, LOW TEMPERATURE	HIGH, LOW WIND IN KNOTS	WEATHER	PRODUCTION HRS.	L.A. LAB TEST & MOD. HRS.	EQPT. BREAK-DOWN & REPAIR HRS.	TIME LOST WEATHER HRS.	AVERAGE DRIVING TIME HRS.	LAYOUT PICKUP TIME HRS.	PAYROLL TIME FOR OBSERVER HRS.	REMARKS
11-6	31/23	8/0	SNOW FLURRIES	0	0	0	0	0	0	10	TRAVEL DAY
11-7	41/24	12/5	RAIN COOL	0	0	9 1/2*	0	1 3/4	0	10 3/4	*SET UP TIME
11-8	51/23	8/0	CLEAR	0	0	9 3/4*	0	1 3/4	0	11	* " " "
11-9	62/41	12/8	"	2	0	3 1/4	0	3 1/4	1	11	LOW VOLTAGE: BOUGHT BATTERY CHARGER
11-10	73/43	8/0	"	2 3/4	0	3	0	3 1/4	1	10	BOUGHT 4 MORE BATTERIES
11-11	63/45	15/8	RAIN	0	0	4	0	0	0	4	T/B NOISE: WORKED AT OFFICE
11-12	47/45	15/6	CLEAR	0	0	0	0	0	1	0	DAY OFF
11-13	37/35	10/5	RAIN	4 1/4	0	7	0	3 1/4	1	15 1/2	NOISE IN T.F.A. AMPS
11-14	33/31	12/2	COOL, OVERCAST	2 1/4	0	8	0	3 1/4	1	14 1/2	CHANGED WIRING IN T/B BACK TO ORIGINAL
11-15	35/26	15/5	RAIN	5 3/4	0	0	0	3 1/4	1	10	GOOD DAY OF PRODUCTION

SUMMARY OF FIELD OPERATIONS

DATE	HIGH, LOW TEMPERATURE	HIGH, LOW WIND IN KNOTS	WEATHER	PRODUCTION HRS.	L.A. IA3 TEST & MOD. HRS.	EQPT. BREAK-DOWN & REPAIR HRS.	TIME LOST WEATHER HRS.	AVERAGE DRIVING TIME HRS.	LAYOUT PICKUP TIME HRS.	PAYROLL TIME FOR OBSERVER HRS.	REMARKS
11-16	38 / 23	8 / 0	OVERCAST	4 3/4	0	1	0	3 1/4	1	10	BAD RELAY
11-17	41 / 38	10 / 0	RAIN	2	0	2 1/2	1/2	3 1/4	1	9	SHORTED POT IN 6 DELCO
11-18	37 / 29	20 / 10	CLOUDY	2 1/2	0	1	0	3 1/2	1	8	NOISY TRANSISTOR IN TFA
11-19	46 / 35	18 / 0	CLEAR	0	0	10	0	0	0	10	CHANGED CLIPS ON PHONE AT MOTEL
11-20	37 / 29	20 / 0	SNOW FLURRIES	0	5 3/4	0	1	1 3/4	1	10	RAN TEST ON PHONES
11-21	47 / 33	0 / 0	SNOW FLURRIES	5 3/4	0	0	0	3 1/4	1	10	
11-22	56 / 48	12 / 0	FOG	4 1/4	0	2	0	3 1/4	1	10½	WET PLUGS: RAIN BLOWING IN BRONCO
11-23	49 / 34	12 / 2	RAIN	0	0	0	5	3 1/4	1	6*	RAIN *LOST TIME DUE TO RAIN
11-24	39 / 25	0 / 0	RAIN SNOW	5 3/4	0	0	0	3 1/4	1	10½	GOOD DAY'S SHOOTING
11-25	12 / 4	49 / 40	OVERCAST	4 1/4	0	1/2	1	3 1/4	1	9½	SHORT IN CABLE PLUG

SUMMARY OF FIELD OPERATIONS

DATE	HIGH, LOW TEMPERATURE	HIGH, LOW WIND IN KNOTS	WEATHER	PRODUCTION	L.A. IA3 TEST & MOD.	EQPT. BREAK-DOWN & REPAIR	TIME LOST WEATHER	AVERAGE DRIVING TIME	LAYOUT PICKUP TIME	PATROLL TIME FOR OBSERVER	
				HRS.	HRS.	HRS.	HRS.	HRS.	HRS.	HRS.	
11-26	12 / 4	49 / 40	OVERCAST	0	0	5	0	0	0	5	ADJ HIGH SPEED TRANSPORT DRUM
11-27	12 / 3	57 / 30	OVERCAST	3 1/4	0	0	2	3 1/4	1	9 1/2	HIGH WINDS TROUBLE WITH PIPS
11-28	18 / 8	45 / 23	OVERCAST	0	7 3/4	0	0	1 1/4	0	9	TWO MEN CLEARING LINE: FILTER BOX CHANGES ORDERED BY L.A. IA3
11-29	20 / 0	34 / 23	CLOUDY	0	0	8 3/4	0	1 1/4	0	10	TRIED TO GET RID OF CROSS FEED CHECKED ON T/3: CHECKED ALIGNMENT
11-30	10 / 0	41 / 28	RAIN, SNOW	0	0	8 3/4*	10	1 1/4	0	10	*HARD RAIN:REPAIRED FO7 BAD BINARY
12-1	5 / 0	27 / 34	FROST CLEAR	0	3*	3 1/2	10	2	1	9 1/2*	TRUCK BOGGED DOWN: CHECKED FILTERS: MOVED T/3
12-2	5 / 0	43 / 24	RAIN	3	0	0	4*	2	1	10 1/2	*WORKED ON HIGHWALL COMPRESSOR FROZEN:CHANGED HF 1.8 FILTERS
12-3	15 / 0	46 / 35	RAIN	0	1	2 1/2	0	0	0	3 1/2	WORK PART OF DAY ON T.F.A. AMPS
12-4	8 / 0	42 / 29	OVERCAST	0	2	7 1/4	0	1 1/4	0	11	HELPERS DRILLED ON HIGHWALL TRIED TO FIND CAUSE OF PIPS
12-5	5 / 0	50 / 43	OVERCAST	6	0	2	0	2	1	11	REPAIRED 6 DELCO: REPAIRED POWER SUPPLY

SUMMARY OF FIELD OPERATIONS

DATE	HIGH, LOW TEMPERATURE	HIGH, LOW WIND IN KNOTS	WEATHER	PRODUCTION HRS.	L.A. LAB TEST & MOD. HRS.	EQPT. BREAK-DOWN & REPAIR HRS.	TIME LOST WEATHER HRS.	AVERAGE DRIVING TIME HRS.	LAYOUT PICKUP TIME HRS.	PAYROLL TIME FOR OBSERVER HRS.	
12-6	12 / 0	56 / 34	OVERCAST	0	0	9 1/2	0	1 1/4	0	10 3/4	REPAIRED SWITCH HEAD BOX RELAY
12-7	10 / 0	61 / 45	HIGH WINDS OVERCAST	0	2 3/4*	2	1 1/2	3 1/4	1	10 1/2	*TRYING TO GET THE CORRECT SIZE CHARGE FOR SHOOTING CDP: REPAIRED BROKEN CAM IN HIGH SPEED TRANSPORT
12-8	15 / 0	54 / 40	CLOUDY	9 1/4	0	4 1/2	0	3 1/4	1	10 1/2	NOISE IN CHANNEL 13-24: REPAIRED SHORT IN CAMERA PLUG: 3 OPEN COLLINS CHOKES
12-9	3 / 0	50 / 26	FOG	0	0	10	0	0	0	10	CLEANED UP RECORDING TRUCK, REPAIRED PHONES, STANDING BY FOR PARTS: 4 MEN DRILLED ON HIGHWALL
12-10	12 / 8	50 / 44	RAIN	0	0	0	0	0	0	0	STANDING BY FOR COLLINS CHOKES
12-11	5 / 0	52 / 45	RAIN	0	0	0	0	0	0	6	INSTALLED COLLINS CHOKES MADE CHECKS
12-12	15 / 0	52 / 45	RAIN	1 1/4	0	3	2*	3 1/4	1	10 1/2	ROADS VERY BAD *VERY HIGH WINDS HEADS NOT SEATING CORRECTLY
12-13	15 / 0	57 / 37	COLD OVERCAST	6 1/4	0	4	0	3 1/4	1	14 1/2	WEAK T/B: WATER IN PATCH PANEL (HIGH CROSSFEED)
12-14	10 / 0	35 / 28	RAIN	2 1/4	0	2	3	3 1/4	1	11 1/2	PROBLEMS WITH POWER SUPPLY: CALLED L.A. FOR NEW INVERTER
12-15	35 / 28	10 / 0	WIND COLD	1/34	0	2	2	3 1/4	1	10	HAVING TROUBLE WITH HEADS WORKED ON TIME BREAK

SUMMARY OF FIELD OPERATIONS

DATE	HIGH, LOW TEMPERATURE	HIGH, LOW WIND IN KNOTS	WEATHER	PRODUCTION HRS	L.A. LAB TEST & MOD. HRS.	EQPT. BREAK-DOWN & REPAIR HRS.	TIME LOST WEATHER HRS.	AVERAGE DRIVING TIME HRS.	LAYOUT PICKUP TIME HRS.	PAYROLL TIME FOR OBSERVER HRS.	
12-16	35 / 23	12 / 0	CLEAR WIND COLD	2 3/4	0	3	3*	3 1/4	1	10	*VERY HIGH WINDS REPAIRED AMPS: ROADS VERY BAD
12-17	50 / 23	3 / 0	STILL & NICE	6 2/4	0	0	0	3 1/4	1	11	STILL HAVING PIP TROUBLES
12-18	66 / 47	5 / 0	WARMER RAIN	6 1/4	0	0	2*	3 1/4	1	12 1/2	*RAIN: FAIR DAY'S SHOOTING
12-19	57 / 40	5 / 0	STILL CLEAR	3	3	1 1/2	0	3 1/4	1	11 1/2	TRYING TO CLEAR UP PIPS
12-20	61 / 34	0 / 0	LIGHT HAZE	2	5	1	0	3 1/4	1	12	
12-21	72 / 52	15 / 5	RAIN WIND	1 1/4	4	0	1 1/2*	3 1/4	1	11	*RAIN
12-22	45 / 29	10 / 5	RAIN WIND	1	0	2 1/4	2 1/2	3 1/4	1	10	PREFILTER OUT WAITING FOR COLLINS CHOKES
12-23	23 / 19	12 / 5	SNOW WIND	0	0	0	0	0	0	0	WAITING FOR COLLINS CHOKES
12-24	? / ?	10 / 0	OVERCAST COLD	0	0	0	0	0	0	0	" " " "
12-25	46 / 34	15 / 0	RAIN	0	0	5	0	0	0	5	STAN INSTALLED & CHECKED COLLINS CHOKES

SUMMARY OF FIELD OPERATIONS

DATE	HIGH, LOW TEMPERATURE	HIGH, LOW WIND IN KNOTS	WEATHER	PRODUCTION HRS.	L.A. LAB TEST & MOD HRS.	EQPT. BREAK-DOWN & REPAIR HRS.	TIME LOST WEATHER HRS.	AVERAGE DRIVING TIME HRS.	LAYOUT PICKUP TIME HRS.	PAYROLL TIME FOR OBSERVER HRS.	
12-26	30/22	12/2	SNOW WIND	0	0	10 1/2	12 1/2*	1 1/2	0	12 1/2	REWORKED PHONES: WORKED ON PIPS: CLEANED TRUCK *NEW SNOW, CAN'T GET UP TO HIGHWALL
12-27	33/16	[illegible]/0	SNOW	3 1/2	0	0	3*	3 1/4	1	11	VERY WET AND COLD PRODUCTION VERY COLD
12-28	33/27	10/0	SNOW	2 3/4	3	1	0	3 1/4	1	11	REASSEMBLED PHONES: TEST FOR CURT JOHNSON
12-29	29/23	10/0	CLEAR COLD	3 3/4	0	0	2*	3 1/4	1	10	* LINE VERY SLICK & ON STEEP HILL
12-30	27/21	5/0	CLEAR COLD	1 2/3	0	6*	0	3 1/4	1	12	SNOWING,WET,COLD: CAMERA BREAKDOWN PIPS BACK STRONG,VERY SLOW PRODUCTION
12-31	?/?	5/0	COLD	1 1/4	0	3	6*	3 1/4	1	12	TRIED TO WORK: VERY SLOW GOING UP & DOWN LINE
1-1	14/6	5/0	VERY COLD	2 1/4	0	0	3*	3 1/4	1	9 1/2	*ROADS VERY BAD LEFT RECORDING TRUCK IN FIELD
1-2	-7/-15	0/0	VERY COLD							10 1/2	EQPT FROZEN: TOLD TO PICKUP LATER TOLD TO TRY AGAIN
1-3	30/24	24/0	NEW SNOW	0	0	0	0	3 1/2	0	3 1/2	CAN'T GET UP THE HILL
1-4	38/9	10/20	WIND SNOW	4 3/4	0	1	2*	3 1/4	1	12	*PHONES FROZEN TO THE GROUND

SUMMARY OF FIELD OPERATIONS

DATE	HIGH, LOW TEMPERATURE	HIGH, LOW WIND IN KNOTS	WEATHER	PRODUCTION HRS.	L.A. LAB TEST & MOD. HRS.	EQPT. BREAK-DOWN & REPAIR HRS.	TIME LOST WEATHER HRS.	AVERAGE DRIVING TIME HRS.	LAYOUT PICKUP TIME HRS.	PAYROLL TIME FOR OBSERVER HRS.	REMARKS
1-4	5 / 8	27 / 20	WIND SNOW	4 3/4	0	1	2*	3 1/4	1	12	*PHONES FROZEN TO THE GROUND
1-5	5 / 0	17 / 2	CLEAR COLD	0	0	0	10 1/2*	3 1/4	1	10 1/2	*HIGH SPEED TRANS. FROST ON HIGH SPEED TRANSPORT: DRILL AMPS OUT FIRE: OPENED PLUGS & DRIED OUT
1-6	12 / 0	31 / 17	SNOW	0	0	0	10*	1 1/4	0	3	NEW SNOW ON GROUND PUT INSTRUMENTS IN HEATED ROOM
1-7	10 / 0	16 / 4	WIND SNOW	0	0	0	10*	1 1/4	0	4	BOUGHT COLEMAN HEATER TO HEAT RECORDING TRUCK: ROADS AND TRAILS VERY SLICK
1-8	15 / 0	+13 / -5	CLEAR COLD	0	0	0	10*	3 1/4	1	10	2½' DRIFTS OF NEW SNOW ON ROAD UP TO LINE
1-9	3 / 0	26 / 9	COLD SNOW FLURRIES	0	0	6	6*	3 1/4	1	10	BIG PIPS BACK ON RECORDS REWORKED GEOPHONES
1-10	12 / 8	28 / 14	OVERCAST COLD	2	0	0	3 3/4	3 1/4	1	10	*BAD CROSSFEED DRIED PATCH BOARD BY FIRE: CLEANED PLUGS WITH CARBON TE+
1-11	8 / 0	32 / 19	WARMER CLEAR	0	0	6 3/4	0	3 1/4	1	11	WORN GEAR IN HIGH SPEED TRANSPORT PULLED & CHANGED, STILL WONT WORK
1-12	8 / 0	28 / 17	CLEAR	1	0	5 3/4	0	3 1/4	1	11	CHANGED HIGH SPEED TO 10 TIMES MOTOR: TOOK 3 SHOTS: TIMING LINE LIGHT OUT
1-13	12 / 2	26 / 16	RAIN SLEET SNOW	4 1/4	0	0	2*	4 3/4º	1	10	º FINISHED LINE #4. SHOT 6 SHOTS ON LINE 1 LONG MOVE AROUND

SUMMARY OF FIELD OPERATIONS

DATE	HIGH, LOW TEMPERATURE	HIGH, LOW WIND IN KNOTS	WEATHER	PRODUCTION HRS.	L.A. LAB TEST & MOD. HRS.	EQPT. BREAK-DOWN & REPAIR HRS.	TIME LOST WEATHER HRS.	AVERAGE DRIVING TIME HRS.	LAYOUT PICKUP TIME HRS.	PATROLL TIME FOR OBSERVER HRS.	
1-14	8 0	32 14	OVERCAST	6 3/4	0	0	2*	3 1/4	1	11	SHOT ON LINE #2 TEST AREA: *VERY SLICK, MEN MOVING SLOW
1-15	5 0	28 17	SNOW	0	0	0	10	1 1/4	0	8	SNOWING HARD MOST OF DAY. TOOK STAN & PUT ON PLANE FOR L.A. AT 3:30 PM
1-16	0 0	26 11	COLD CLEAR	2	0	0	0	3 1/4	1	11	MEN DRILLED 10 HOLES FOR HIGHWALL: SHOT HIGHWALL: CLEARING & DRILLING TIME 7 HRS.
1-17	12 2	32 19	CLEAR	0	0	0	0	1 1/4	0	10	END OF PRODUCTION: PACKING EQUIPMENT
1-18	8 0	38 23	OVERCAST	0	0	0	0	3 1/2	0	8	PACKING EQPT. TOOK ONE BRONCO TO WILSON MOTORS
1-19	10 0	36 27	CLEAR WARMER	0	0	0	0	3 1/2	0	P	FINISHED PACKING & DELIVERED TO TRUCK LINE: TOOK 2nd BRONCO TO WILSON MOTOR
1-20	5 0	44 29	CLEAR	0	0	0	0	0	0	P	
1-21	5 0	36 24	CLEAR	0	0	0	0	0	0		DAY OF REST
1-22	6 0	29 17	SNOW	0	0	0	0	0	0		CHECKED FINAL TIME WITH BYRON CONST. CHECKED BRONCOS AT WILSON MOTORS CLOSED BANK ACCOUNT
TOTALS				138	38	187.5	143.75	181	48	685	AT 6:35 DROVE TO PITTSBURG TOOK 8:40 PLANE FOR L.A.